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DEPARTMENT OF THE INTERIOR
ALBERT B. FALL, SECRETARY
BUREAU OF MINES
H. FOSTER BAIN, DIRECTOR

TESTS OF LOW-GRADE AND COMPLEX ORES IN COLORADO

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BY

WILL H. COGHILL and C. O. ANDERSON

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TESTS OF LOW-GRADE AND COMPLEX ORES IN COLORADO.

By Will H. Coghill and C. O. Anderson

INTRODUCTION.

PURPOSE AND SCOPE OF THE INVESTIGATION.

Those familiar with the mining industry in Colorado have recognized that metal mining has been on the wane throughout the State for several years and that the mineral production has decreased in both quantity and value, until metal mining no longer holds the leading place among the industries of the State.

Statistics recently compiled by the State board of immigration[1] from Government, State, and other sources, show the relative importance of the basic industries of the State according to the value of their products as follows: Agriculture, live stock, manufacturing, and mining. It will be noted that mining has been given fourth place, which represents a marked decline from the premier position formerly held.

The reason metal mining has failed to maintain its former position of importance is twofold; first, the other industries of the State, particularly agriculture, have had an unusually rapid growth in recent years, which has greatly increased the value of their output; and, second, the mineral production of the State has decreased because the rich and more readily accessible deposits have been largely exhausted, and the primary sulphide ores are in many instances too low grade to be profitably exploited or too complex in their mineralogical and physical composition to be amenable to the usual treatment methods for turning the valuable metallic constituents to commercial account.

Naturally the mining men of the State are much concerned over the future of the industry, and numerous suggestions have been advanced from time to time as to steps that might be taken to rejuvenate mining and thus restore to it something of its former position of importance.

[1] Colorado Year Book, 1920.

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As a first step in this direction it was proposed that a field investigation of the low-grade and complex ores of the State should be undertaken jointly by the Colorado School of Mines and the United States Bureau of Mines, with the object of determining the character and approximate quantity of these ores, and, where found in sufficient quantity to constitute a metallurgical problem, to carry on experimentation at the laboratories in Golden in an effort to devise suitable treatment, and thus make them commercially valuable. The officials of the School of Mines and of the Bureau of Mines readily gave their consent to this proposal and as a result a bill, providing for a joint investigation, was introduced by Messrs. Hall, Girard, and Jenkins at the special session of the legislature, which was called by Governor Shoup in December, 1919.

The bill as passed by both branches of the legislature and approved by the governor on December 19, 1919, is known as house bill No. 13, and is as follows:

An act to create a fund known as State ore testing and experimentation fund, to encourage, assist, and promote the metal mining industry and providing the manner in which such fund shall be used.

Be it enacted by the General Assembly of the State of Colorado:

SECTION 1. For the purpose of encouraging and assisting the mining industry of the State of Colorado and to promote the extension of such industry there is hereby appropriated for the calendar year 1920, out of any money in the treasury not otherwise appropriated, the sum of fifteen thousand dollars ($15,000), which amount shall constitute a fund to be known as the State ore testing and experimentation fund. Such fund shall be used only for the purpose of testing and experimenting in the treatment and reduction of Colorado ores, under the supervision and direction of the board of trustees of the Colorado State School of Mines in furtherance of the plans and along lines determined by the United States Bureau of Mines.

SEC. 2. The appropriation herein provided shall be an appropriation of the third class.

SEC. 3. The auditor of the State is hereby authorized and directed to issue his warrants against the State ore testing and experimentation fund in payment of expenses incurred in ore testing experiments and investigations made under the supervision of the board of trustees of the Colorado State School of Mines, upon vouchers signed by the president and secretary of such board of trustees and approved by the State auditing board.

SEC. 4. In the opinion of the general assembly an emergency exists; therefore, this act shall take effect and be in force from and after its passage.

FIELD WORK.

Preliminary plans for the State survey and experimental work were made after consultation with a group of prominent mining engineers, representing, as far as possible, the various mining districts of the State. In forming the program for the work the Bureau of Mines was represented by D. A. Lyon, chief metallurgist and supervisor of stations; by Dr. R. B. Moore, chief chemist; by Dr. S. C. Lind,

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superintendent of the Golden station; and by Will H. Coghill, metallurgist. The School of Mines was represented by Dr. Victor C. Alderson, president; by J. C. Williams, assistant director of the experimental plant; and by several members of the faculty of the School of Mines. The general direction of the work was placed in the hands of the superintendent of the Golden station of the Bureau of Mines, in consultation with the president of the School of Mines.

The work was, in general, divided into two parts, consisting of a field survey of the principal mining districts and of metallurgical experimentation upon the samples forwarded from the field to the laboratories at Golden.

R. R. Hornor, mining engineer of the Bureau of Mines, was furloughed to take charge of the field work, with B. C. Essig representing the State as his assistant. Mr. Essig took up the work about the middle of February, 1920. During February and March the old mining districts of Georgetown, Silver Plume, Idaho Springs, and Central City were visited, and later Breckenridge, Leadville, and Aspen.

On April 15, Mr. Hornor took up the field work and in company with Mr. Essig made a more detailed study of the ores of Gilpin and Clear Creek Counties. The next districts visited were Creede, Del Norte, Bonanza, Ohio City, Whitepine, and Monarch. Finally, the San Juan region, including the Silverton, Eureka, Red Mountain, Ouray, Telluride, Rico, and Lake City districts, was investigated during July and August. This, with a further visit to Leadville and Kokomo, completed the field survey.

In each of the districts visited a careful investigation was made of the mining and milling practice, character of the ores being mined and milled, and the treatment problems involved. Altogether 63 metallurgical samples were obtained from the various ore deposits examined, and forwarded to Golden for examination and treatment.

EXPERIMENTAL WORK.

The experimental work consisted, first, in a careful physical examination of the ore, followed by a complete chemical analysis to determine the valuable metallic constituents. If these preliminary tests showed that the combined minerals were so locked in the ore that they could not be readily separated into marketable products, then methods of treatment were applied which experience had shown would most likely give the desired results. The metallurgical work was under the immediate direction of Will H. Coghill, assisted by C. O. Anderson, J. P. Bonardi, and Max Shapiro, of the United States Bureau of Mines, and by J. C. Williams and E. V. Ingels, representing the State.

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ACKNOWLEDGMENTS.

The writers take this opportunity to show their appreciation and to thank the mine owners, operators, managers, superintendents, engineers, and mining men in general of the State for the many courtesies extended and for supplying much valuable information; also for permission granted and facilities offered to inspect the mines and mills in the various districts visited, and for samples of ore furnished.

Special acknowledgment is due to Horace F. Lunt, State commissioner of mines, whose helpful suggestions have done much to facilitate the work in the field; to M. B. Tomblin, secretary of the Colorado Metal Mining Association, for many favors and keen interest shown in the work; and to the Hon. J. G. Slattery, State senator, Silverton, Colo., who was exceedingly active in his efforts to obtain the appropriation to carry on this work, and later gave Messrs. Hornor and Essig every assistance possible when they were engaged in field investigations in the Silverton district.

MINERAL AREAS OF COLORADO.

A glance at the accompanying map of Colorado (Fig. 1) will show that the mineral areas are confined to the mountainous parts of the State, and that within this region the ore deposits are not by any means uniformly distributed. The most important deposits are confined mainly to a diagonal belt extending from Boulder County southwest to Dolores County—a distance in a direct line of nearly 250 miles.

This main mineral belt may be divided into three areas, which for convenience will be designated as follows:

(1) Northeastern, comprising the mining districts in Boulder, Gilpin, Clear Creek, Summit, Eagle, Park, and Lake Counties.

(2) Central, comprising the mining districts in Pitkin, Chaffee, Gunnison, and Saguache Counties.

(3) Southwestern, comprising the mining districts in Ouray, San Miguel, Dolores, San Juan, Hinsdale, and Mineral Counties, commonly known as the San Juan region.

Outside the main mineral belt and lying in the front range of mountains are the districts of Cripple Creek in Teller County and Silver Cliff in Custer County. Cripple Creek is well known as one of the principal gold-producing districts of the United States, and Silver Cliff was formerly an important producer of rich silver ores.

Besides the gold and silver deposits in the districts mentioned above, there is an entirely different class of deposits situated mainly in Montrose and San Miguel Counties. These deposits contain ores of vanadium, uranium, radium, and occasionally copper, disseminated in sedimentary beds, chiefly sandstone.

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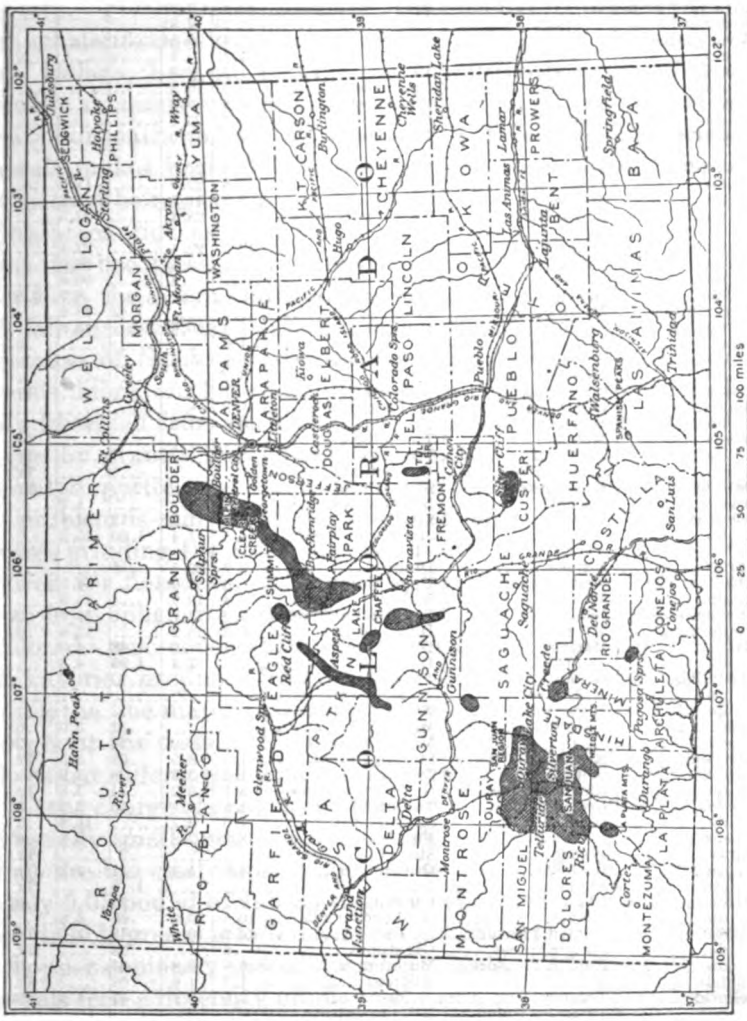

FIG. 1.—Map of Colorado showing approximate distribution of principal lead, silver, and gold regions.

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TABLE 1.—*Sample record.*

No.	Camp.	Mine.	Location of samples.	Analysis.				Assay (per ton).	
				Pb.	Zn.	Cu.	Fe.	Au.	Ag.
				P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*Ozs.*	*Ozs.*
1	(*a*) Silver Plume.			23.20	3.27				22.75
	(*b*) Silver Plume.	Mendota	Fourth level	10.23	2.53				6.80
2	Silver Plume	do	Third level	1.48	5.54				3.65
3	do	Dives-Pelican	Dump	1.1	.3				14.04
4	Georgetown	Colorado Central	Mill bins	.3	2.2				2.07
5	do	Griffith	Tunnel level	.6	10.3				2.34
6	Silver Plume	Mendota	Third level	2.4	16.3				5.04
7	do	Smuggler	Fifth level	18.1	25.4				121.03
8	Georgetown	Georgetown T. & T. Co.	Tunnel level	17.3	12.0				150.93
9	Leadville	Down Town	Eighth level	.3	20.0				.39
10	do	Ponsardin	Third level	15.6	22.0		11.6		2.56
11	do	Whitecap	No. 27 raise	2.4	11.2		28.3		.79
12	do	South Mike	100 feet northeast of shaft.		28.4		23.5		4.99
13	do	South Moyer	Ore bin	12.2	22.4		22.4		5.67
14	do	Ibex	Seventh level		2.3		36.6		3.28
15	do	do	Sixth level		.4		56.0		
16	do	Garbutt	Cortellini stope				8.7	0.89	
17	Breckenridge	Molly B	Tunnel level			1.42	18.6		.92
18	do	Wellington	Mill feed	.8	20.6		43.8		3.8
20	Aspen	Little Annie	do	1.0				.01	11.93
21	do	Smuggler	No. 2 tunnel	6.2	31.8				7.48
22	do	Newman	Ore bin	3.2					3.36
23	Bonanza	Cocomongo	Mill feed	.7				Tr.	2.62
24	Central City	Gilpin-Eureka	Zn-Fe middling	8.8	8.0				4.90
25	do	Iron City Mill	do	10.6	11.5		27.2		7.64
32	Creede	Monon	Tunnel level					Tr.	13.20
33	do	Manitoba	First level					Tr.	11.66
34	do	Commodore	No. 3 tunnel					Tr.	13.24
35	Del Norte	Esperanza	200 feet level					Tr.	21.00
36	Creede	Bachelor	No. 4 tunnel				1.7		7.75
37	Bonanza	Rawley	Zinc vein	6.8	11.7	.12	11.8		1.46
38	do	do	Sixth to twelfth levels.	1.3	1.4	1.60	15.3		5.90
39	Garfield	Eureka	Dump	5.7	20.5	.16			2.0
40	Kokomo	Michigan	Upper tunnel	.6	13.0		34.3		1.02
41	do	Wilfley	Mill	.4	10.9		23.6		.88
42	Breckenridge	Jessie	Fe-Zn concentrate	1.0	17.7		31.4	.27	2.25
43	Montezuma	Morgan	Dump, lower	6.2	18.2		13.6	.02	8.72
44	Rye	Marion	Dump, upper		.6	1.68	4.0		
45	do	do	Dump		7.3	1.96	4.8		
46	Leadville	Griffin	Tunnel level				11.7		
47	do	R. E. Lee	235-foot level						9.74
48	do	Duncan No. 4	160-foot level						5.86
49	Alice	Alice Placer	Mill feed			.12	5.0	.01	.08
50	do	do	Mill concentrate			1.24		.21	1.46
51	Silverton	Richmond-Sterling	Sterling tunnel	11.5	3.5	.08	3.8	Tr.	2.72
52	do	Columbus	Dump	1.8	3.6	.28	2.2	Tr.	1.24
53	do	Richmond-Sterling	Garry Owen tunnel	3.1	6.2	.34	3.8	Tr.	.54
54	do	Iowa Tiger	Table feed	1.8	2.8	.56	3.2	.07	1.03
55	do	Gold King	Second level	1.6	2.7		16.4	.32	11.40
56	do	do	First level	3.8	7.8		6.8	.05	4.33
57	Ouray	Wedge	Stope fill			.2		Tr.	24.84
58	do	Bachelor	Ore bin	4.6	5.4	.24	1.8	Tr.	13.80
59	do	Senorita	Dump			1.16		Tr.	25.04
60	do	Black Girl	Ore bin			.96		.03	21.57
61	do	Beaver	Gold Crown mill	10.6	20.0	.66	6.6		1.52
62	do	Hidden Treasure	do	14.2	10.6	.48	3.7	Tr.	5.96
63	do	Pony Express	Old stopes	.6		.08		Tr.	18.26
64	do	Hidden Treasure	Mine dump	6.8	6.4	.44		.02	10.9
65	Sneffels	Atlas	Mill feed	1.3	1.6	.08		Tr.	7.34
66	Telluride	Tomboy	do	2.6	3.7	.52	4.5	.32	3.00
67	do	Smuggler-Union	Mill concentrate	(1)	(1)	(1)	(1)	(1)	(1)
68	do	Black Bear	Mill feed	2.3	4.2	1.04	4.6	.11	2.12
69	Rico	Rico Consolidated	Intermediate tunnel	11.4	13.8	.38	16.8		4.92
70	do	Rico-Argentine	Lower dump	14.8	22.0	1.0	10.7		4.36

[1] Not reported.

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TREATMENT OF THE ORES.

COMMENTS ON COMMERCIAL AND LABORATORY PRACTICE.

Differential flotation—that is, flotation of one mineral in advance of another to make a complete separation—is not generally successful, and, with one exception, its application to Colorado lead-zinc ores is a failure. Fortunately, however, the success of separating galena from sphalerite does not stand or fall with so-called differential flotation. Galena, because it has a greater specific gravity than the economic minerals accompanying it, is well adapted to recovery by gravity concentration. Only when intimate interlocking with other minerals makes impossible a physical process of separation should this method be neglected.

Due precaution as to the manner of grinding is necessary. The grains that become excessively ground may require flotation to recover them from the gangue minerals, but, in turn, they may be recovered with fair success from the flotation overflow by concentrating tables.

Because of the lesser specific gravity of sphalerite, its recovery depends more on flotation than does galena. However, if it is easily liberated from its matrix, it, too, may be recovered from the gangue by gravity concentration. Separation from the commonly associated pyrite may then be made by magnetic concentration. The problem is more complicated where the nature of the mill feed requires grinding the sphalerite and pyrite to a fine mesh such as is required for flotation. Then the customary methods for removing pyrite from sphalerite are more difficult.

Colorado zinc concentrates are obliged to compete in the market with superior products from other sources and a better process for grading the fine material is imperative. This need was kept in mind throughout the tests.

So-called differential flotation is generally dependent on the addition to the charge of certain carefully selected reagents in very definite and usually small quantities. In such a mixture the favorable conditions are too easily upset; rock drills spray into the stope approximately 0.05 pound of lubricating oil per ton of ore; aerial tramway ropes dribble grease into the loaded buckets, and crushers placed over the hopper copiously spread their lubricants. All these at irregular intervals feed a diversity of oils to the ore, and are likely to interfere with the effect of the reagent chosen. Gravity concentration will often aid in combating this menace.

Metallurgists have plunged into the practice of flotation with the idea that very fine grinding is not objectionable and that the economic minerals can be floated successively to make satisfactory concentrates of each; as a result types and arrangements of grinding

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machinery have been adopted that are hard to remodel and adapt to gravity concentration for the recovery of galena. Because of the longevity of stamps they still exist in many of the mills and are offenders in causing excessive fineness. The difficulty of recovering galena is increased by too fine grinding, but if the grinding is properly performed the losses will be moderate. Stage grinding is imperative.

Classifiers, so well developed in and so well adapted to the cyanide process, are found working in closed circuit with the grinding machines and as a result the sulphides are ground too much. The classifier discharge is usually so regulated that locked sulphides are kept in the circuit until liberated, and consequently free sulphides hydraulically classified are retained in the grinding circuit and ground too fine for the most efficient gravity concentration. This is very noticeable when the heavy sulphide, galena, is present. If the economic minerals had a lower specific gravity than the gangue hydraulic classification would be ideal, but the reverse is true, and an extra pass through a grinding mill makes them too fine for good gravity concentration and for smelting.

The necessity of passing the classifier sand over concentrating tables to recover the free sulphides is now being recognized, or the classifier sand may be screened and the undersize passed over a table to recover free sulphides. The last procedure brings up the question, what is the use of a classifier in a closed circuit if a screen will do the work more effectively? The efficacy of the classifier should be seriously questioned when gravity concentration follows. The classifier (in conjunction with the screen) will remove the material that would pass the screen most easily, rob the screen of water required for a wash, and necessitate a dilution in the overflow that will be troublesome in the subsequent treatment.

A rapidly vibrating screen or a traveling screen should be considered as a substitute for classifiers in the closed grinding circuits where table concentration follows. Manufacturers should be encouraged to develop fine-mesh screens.

The cyanide process passed through its era of development without experiencing the successive applications of blundering empiricism suffered by sulphide concentration. Cyaniding was introduced more auspiciously. It is a solution process, and dissolving by means of chemicals was not new; only the application was new. Chemicals were used quantitatively in accord with the common practice of chemistry, and their effects were determined in like manner. Flotation, on the other hand, came as if from another world. It had no relative, and no analogies except, perchance, the "floating needle" and the sulphide held by film suspension. Empiricism was a natural consequence.

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Concentration mills are built without due consideration of the nature of the ore, and the flow sheets usually are not flexible. It is unfortunate that the industry is not better supplied with adequate commercial ore-treating laboratories to determine suitable methods of treatments. For this condition the industry itself is to blame. Too many mining companies prefer to plunge into the purchase of milling machinery and hesitate to spend small sums for ore testing. As a result, the operators of ore-testing laboratories are discouraged from buying more than the most simple equipment. If they were properly supported, the industry would be given better service in ore testing.

In flotation too much is expected of the rougher. To try to recover all the economic mineral in a marketable concentrate and in the same procedure to try to make a low-grade tailing is impractical, that is, the rougher should be used primarily to remove the economic minerals from the gangue and to discharge a low-grade tailing. After the gangue and the water polluted by the flotation reagents have been discarded, the economic minerals may be treated in a separate circuit.

An adequate miniature mill is the only means by which laboratory results may be satisfactorily coordinated with commercial ore treatment. Such a mill should be provided with jaw crushers and rolls arranged for stage crushing, tube mills to grind in closed circuit, tables and multiple-cell flotation machines. It should be designed to treat about 400 pounds a day. Special apparatus for the addition of flotation reagents would be required. In experienced hands, this equipment should give results closely paralleling good milling practice.

Unfortunately, however, the experimenter is generally obliged to work by steps—that is, to apply an intermittent treatment. This is very difficult throughout. During stage grinding in a laboratory ball-mill, the mill should be unloaded at frequent intervals to remove the portion of the charge that has been ground to the desired fineness. When table concentration is desired, much difficulty is caused by the smallness of the table and by the small quantity of feed; laboratory slime tables are almost unheard of.

A great deal of study is required to develop a system of locked testing so that a proposed mill flow sheet may be approximated. In flotation testing the first requisite is to determine how to make a low-grade tailing. This having been done, the low-grade concentrate may be returned for cleaning. The cleaning treatment will produce a middling that must receive careful attention. The binocular microscope is indispensable throughout the work, but it must not be relied on wholly, and guessing about the recoveries that the middlings will yield is to stop short of the proof. The

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middling should be returned to the rougher circuit in a locked test. After several runs, the distribution of the minerals will be evident from the grade of the products.

For many ores, the most desirable procedure in laboratory and commercial flotation is bulk flotation—that is, to float all the economic minerals, and subsequently discharge the tailings. The gangue and the accompanying polluted water having been discarded, the mixed sulphide concentrate may then receive intensive treatment.

Bulk flotation has often been opposed by the argument that a mineral grain once floated will be floated again, but this argument has not been proved valid in the work discussed in this report. Treatment of the mixed sulphide concentrate after discarding the tailing has much to commend it. If the ratio of concentration is five into one, then 80 per cent of the mill feed is at once discarded. Furthermore, polluted water is discarded, and then water carrying the desired chemical reagents may be applied in exact quantities to the mixed sulphides. Too much emphasis can not be placed on this principle. Physical chemists have often demurred when asked to discuss the principles of flotation, because of the multiplicity of solid and dissolved constituents in the mixture. Bulk flotation followed by a separate treatment of the mixed sulphides divides the study into steps so that the conditions may be determined more easily. The conditions favorable to flotation having been thus determined, they may be regained more easily when lost by unavoidable occurrences.

Many ores are naturally in an ideal condition for bulk flotation. There is danger of abusing such ores and causing undue loss, when reagents are added to prevent a certain economic mineral from floating while another is recovered This, however, is the tendency in the much-flaunted differential flotation. If the natural condition of the ore allows one economic mineral to enter the overflow in advance of another, it would be folly not to take advantage of the natural segregation, rather than to add a reagent to make one of the minerals immune to flotation, and, in turn, to add another reagent to nullify the effects of the first reagent. Such practice unduly pollutes the mill solution. To be sure, if no other means will give the desired results, the method must be tolerated, but investigators should strive to eliminate such inconsistencies.

The mixed sulphides resulting from bulk flotation are in excellent condition to yield a high-grade galena concentrate on tables. This statement presupposes adequate stage crushing and the recovery on jigs and tables of galena too coarse for flotation. The capacity of the gravity-concentration unit is increased many fold when the unit treats only the mixed sulphides instead of the mill feed The lead concentrate will be of higher grade because of the richer table feed

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and the previous elimination of siliceous slime; the treatment may be more intensive than when treating mill feed. The excessively ground galena will overflow the table and with other minerals in a similar condition pass off in suspension. Fortunately, however, galena which is fine enough to remain in suspension on the concentrating table will generally float freely in advance of the other minerals that would contaminate a lead flotation-concentrate. The proper application of flotation reagents will then recover the finest galena on the first cell of the rougher. The galena in the rest of the rougher overflow may be recovered by gravity concentration. If sphalerite and pyrite are present in the ore, the tailing from the concentrating table will contain these minerals in a highly segregated mixture. An intensive treatment of this mixture by flotation may now proceed. The scheme adopted must be one that will float the zinc and discharge the pyrite and other minerals as a by-product in the underflow. It will be seen at once that the pyrite that floated in the rougher, having undergone segregation on the table and during the subsequent flotation treatment, will predominate in the flotation underflow and may be regarded as an iron by-product to be thrown away or shipped, depending on its lead, gold, and silver content. This by-product will unavoidably carry some "insoluble," and if desired may be passed over a concentrating table for further grading-up.

PRELIMINARY STUDY OF DICHROMATE SOLUTIONS.

At the outset of the work the literature on flotation was searched for instances of reagents used to effect separation among any of the following four sulphides: Chalcopyrite, galena, sphalerite, and pyrite. To be sure, the economic requirements in Colorado bring the separations down to this problem—to get the copper, lead, and iron away from the zinc. It has been found in general that if these three metals are separated from the zinc, the precious metals, gold and silver, will also be separated.

A study of the action of the various salts used to effect a separation between lead and zinc sulphides by flotation and experimentation with a number of them in this work has led to the following general conclusion: No inorganic salt alone possesses the requisite property of making a decided separation between two sulphides, but almost any salt has at least a slight tendency to effect a separation of two sulphides. In the light of the present knowledge of flotation, however, these slight tendencies can not be controlled enough to effect the complete separations. Certain oils are better for floating one sulphide than for floating another, but no oil is known that will give a sharp separation of two sulphides with good extractions.

Much of the work done in the past on differential flotation has relied partly on the relative degree of flotability of the various sul-

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phides. In general, if an ore containing the four above-mentioned sulphides is floated in neutral water with an oil such as wood oil but with no other reagents, the lead and copper will predominate at the beginning of the run; as the run proceeds the proportion of them in the froth will decrease and the zinc will predominate. As the zinc decreases or as the end of the run approaches, the iron will predominate, and at the end of the run the gangue or sand will constitute most of the overflow. A run cut short will often leave a large percentage of the iron in the underflow and will recover a fair percentage of the other minerals. Some operators, in their efforts to get a zinc product low in iron, have been compelled to cut the run short and consequently leave too high a percentage of zinc in the tailings.

After experiments for some time with various reagents (keeping the number of oils down to two or three) to separate lead and iron from zinc, the following idea was adopted for further work: A reagent must have considerable chemical activity in order to make a decided separation of two sulphides, as very dilute solutions are used in flotation work. Naturally this limits the reagents chiefly to oxidizing agents. The concepts following summarize in general the knowledge gained from the experience with Colorado ores. Oxidizing agents are considered the most likely to effect a separation of two sulphides. If the reagent attacks all the sulphides, those sulphides forming compounds soluble in the flotation medium will be more likely to float, whereas those forming insoluble compounds, such as oxide films, will be rendered immune to flotation.

The various sulphides consume oxidizing reagents at different rates. One sulphide may consume a reagent so rapidly that another sulphide will not be affected and yet will float. An acid solution of a dichromate, such as that of sodium or potassium, has been the only reagent so far found effective in the work of separating the various sulphides in these Colorado ores.[2] Experiments with this reagent will be described, because they represent the procedure used in investigating the various reagents and because this is the only reagent used in making a clean separation of two sulphides. A convenient way of determining the effect of oxidizing agents on the various sulphides in a flotation mixture is to use the iodometric method, and titrate with sodium thiosulphate.[3] These tests were made with pure speci-

[2] There are two United States patents on the use of dichromate solutions; 1,102,738 by Greenway and Lowry, 1,142,821 by Lavers. The first is for neutral and second is for alkaline solutions. The acid solution seems not to be patentable. Thornberry and Mann in their bulletin on "The effects of addition agents in flotation," published at the Missouri School of Mines, November, 1919, show that chromate and dichromate salts and chromic acid have a very retarding effect on the flotation of lead.

[3] The procedure for titrating the dichromate solution with sodium thiosulphate is as follows: A portion of the flotation solution containing the dichromate is filtered and made distinctly acid. About 1 gram potassium iodide (10 per cent solution) is added, followed by starch solution giving the characteristic blue color. Thiosulphate is run in from a burette until the blue color is just discharged.

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mens of the various sulphides ground through 60-mesh. Theoretically, the following equation indicates the amount of acid necessary to liberate the total oxidizing power of the dichromate:

$$\underset{294}{K_2Cr_2O_7} + \underset{392}{4H_2SO_4} \rightarrow K_2SO_4 + Cr_2(SO_4)_3 + 4H_2O + 3/O,$$

Or 392 grams of acid are necessary for 294 grams of dichromate, or 0.73 c. c. of acid (sp. gr. 1.84) is required for 1 gram of dichromate of potassium; 0.81 c. c. acid is required for 1 gram of sodium dichromate. Potassium dichromate was used in the preliminary tests but sodium dichromate was found to be effective in the flotation work and was used because it is cheaper.

To determine the effect of the reagent on sphalerite, galena, and pyrite, 25 grams of the mineral was placed in 300 c. c. of water containing 1 gram $K_2Cr_2O_7$ and 0.73 c. c. H_2SO_4. The mixtures were agitated for various periods of time, then a small portion was filtered and titrated with thiosulphate.

Table 2 shows the results obtained:

TABLE 2.—*Consumption of acid dichromate by sphalerite, galena, and pyrite.*

Period of agitation, minutes.	Acid dichromate consumed, c. c.		
	Sphalerite.	Galena.	Pyrite.
0	0.0	0.00	0.00
2	...	.97	....
5	.0	.97	.45
10	.0	.97	.58
15	.0	....	.75
20	.0	....	....
30	.0	....	.97

Seemingly the sphalerite does not consume the available oxygen in the acid dichromate solution. The surface of the sphalerite did not seem to be affected. The galena readily consumed the available oxygen from the acid dichromate and turned yellow. No quantitative tests were made to find out how much oxygen the galena would consume, but it was noted that 25 grams would consume even the oxygen from 4 or 5 grams of dichromate in a similar test. The pyrite consumes the oxygen from the dichromate gradually and becomes tarnished, so that it has the appearance of chalcopyrite.

Here are three sulphides all showing different rates of consumption of dichromate; the zinc consuming none, the iron having a gradual rate of consumption, and the lead producing an almost instantaneous consumption.

The mixture containing the sphalerite was filtered. The filtrate was tested for zinc and found to contain 0.082 gram. In turn,

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sphalerite (25 grams) was agitated in 300 c. c. water containing 1 gram of acid-free $K_2Cr_2O_7$. The filtrate contained only 0.004 gram zinc, indicating that the acid present in the first test had attacked the zinc appreciably. Furthermore, when this amount of sphalerite was agitated in 300 c. c. water containing 0.73 c. c. H_2SO_4, the filtrate contained 0.302 gram zinc, a far greater amount than when the dichromate was present with the acid.

The filtrate from the galena contained lead. The galena residue was washed in a 2 per cent ammonium acetate solution to dissolve any lead sulphate or oxides of lead. The solution was filtered and 0.2626 gram of lead was found therein. The filtrate from the pyrite contained much dissolved iron. When a synthetic ore charge containing these three sulphides with quartz was agitated in an acid dichromate solution in a flotation machine with oil, the sphalerite floated readily, whereas the bulk of the galena and pyrite remained in the underflow.

Experiments were made in which the amount of acid present per gram of dichromate was varied from an assumed optimum. These showed that the lead and iron sulphides were more readily floated both with increase and decrease of acid. Furthermore, the amount of dichromate consumed (or reduced) was less when the amount of acid present decreased.

The explanation offered is this: The dichromate reacted with the galena, discoloring it and forming some compound on the surface of the particles, inhibiting flotation. This compound became more soluble in the flotation medium as the amount of acid was increased, and, since it dissolved, the galena particle would still float; but when no acid was present the effect of the chemical reaction was lost. As mentioned above, there was lead in the filtrate, an indication that some had been dissolved. As the acid was decreased to the optimum the compound formed on the galena particles was less readily removed and consequently the particle was less floatable. This result is in accord with the curves shown in Figure 2, page 19.

The same holds true for the pyrite. The dichromate solution tarnishes it, but excess acid present will remove the compound composing this tarnish and yet allow the pyrite to float. A quantity of pyrite that had been subjected to the action of the acid dichromate was washed free from this solution and put with quartz into a flotation machine. When oil was added, very little pyrite floated, but when 5 c. c. acid was put into the machine, fully 90 per cent of the pyrite floated. Examination of the floated pyrite under a microscope showed that the tarnish had disappeared.

A small amount of acid is necessary for the dichromate to react on the lead and iron sulphides. In this work it was found that 0.10 c. c. to 0.15 c. c. acid per gram of dichromate works most satisfac-

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torily. There is considerable question as to what compounds are formed by the dichromate with the sulphides. The galena is colored yellow, perhaps forming a lead chromate. However, if galena is subjected repeatedly to fresh acid dichromate solution, it becomes covered with a snow-white deposit of sulphur, possibly formed by the action of dichromate on hydrogen sulphide resulting from the action of the acid on the sulphide. In flotation tests ordinarily the lead simply loses its luster and becomes dull. A decided tarnish on the pyrite has been noted, but no surmise is made as to what compound is formed. There is room for considerable experimental work to settle these questions.

In flotation testing, samples of the solution may be taken frequently for titration to determine how much dichromate is being consumed. To what extent exhausted or partly exhausted dichromate solution may be used is still an open question. The problem of the effect of the accumulation of salts and oil in this solution will require more investigation before it is settled.

In the work on Colorado ores the dichromate was not relied on to separate all the lead from the zinc, the lead being removed as much as possible by other means. The main function of the dichromate was to remove the iron from the zinc, and in doing this it also removed some of the lead, silver, and gold from the zinc. All types of Colorado zinc sulphide so far studied have been found to float readily in this dichromate solution, although the types containing combined iron are less floatable than those free from iron. The floatability of iron-bearing zinc sulphides in dichromate solution may be increased by finer grinding and by grinding in oil.

After the completion of the preliminary experimental work with acid dichromate solutions as just described, it was desirable to carry out some tests with this reagent in the laboratory flotation-machines. "Made-up" or synthetic charges of the various sulphides with a siliceous gangue were tried first.

In the early part of the work particular attention was paid to the use of dichromate for separating lead and zinc and only incidental attention was given to the fact that the pyrite was retarded. Late in the testing, when it had been decided to remove the lead as much as possible by other means, the main importance attached to the dichromate was its function in removing pyrite from the sphalerite or, in other words, in aiding to "grade-up" the zinc to a desirable commercial product. Another point in this early work was the stress laid upon the treatment of the three sulphides, galena, sphalerite, and pyrite. It was thought best to attempt first to work out a system of treatment for these three minerals and later to give attention to the chalcopyrite (which, as a rule, occurs in Colorado ores in quantities

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equivalent to not more than 0.50 per cent copper) and to the precious metals, gold and silver.

Bulk flotation and locked testing were emphasized in the "comments," but in the first tests most of the work was done by single runs with 500-gram or 1,000-gram charges, in order that the general behavior of the ore and reagents might be easily determined. With the laboratory facilities, results could be obtained from a small charge of ore in a short time, whereas for a locked test of say 10,000 grams, the results would not be complete for four or five days. Locked testing of large charges of ore before certain essential conditions are known is almost entirely a waste of time.

Terms for the various steps in the treatment in the dichromate solution must be assigned. It has been stated that the treatment of a sphalerite-pyrite mixture in the dichromate solution gives an iron product, or by-product, in the underflow from the flotation machine. Such a product is then a "tailing," or, at least, it is a finished product as far as flotation is concerned. As the treatment makes a tailing, or the equivalent of a tailing, it must necessarily be "roughing flotation treatment." Any subsequent treatment of the froth in the solution is a "cleaning" treatment, because middlings are made and returned to the circuit. Furthermore, in locked testing, the running of each charge constitutes a "run." Thus, a middling made in the third run will be added to the fourth run. The entire operation, including gravity concentration and flotation, constitutes a "locked test."

The flotation machines mostly used are the 4-liter and the 9-liter hollow-shaft type. The concentrating table is a 12 by 30 inch Wilfley. Fine grinding was done by means of flint pebbles in porcelain jars, unless otherwise stated. City water was used; after boiling, it reacted alkaline to phenolphthalein, indicating the presence of bicarbonates.

The analytical work was done with accuracy and speed by E. V. Ingels.

In some of the tables where the error is allowable, though appreciable, the feed assay is "calculated."

The terms, selective flotation, preferential flotation, and differential flotation, have no place in the following report. They are unnecessary. There are only two procedures that require naming. (1) The economic minerals are floated together. If no further separation is desired, the term "flotation" is entirely adequate. If subsequent separation is necessary, the term, bulk flotation, may be used in place of "flotation." (2) If one mineral is floated in advance of another, it is better to say so rather than to befuddle the reader with meaningless terms.

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TESTS OF COMPLEX SULPHIDE ORES.

SMUGGLER, AT SILVER PLUME.

The field sample was taken from the fifth level of the Smuggler mine, at Silver Plume, operated by the Stewart Marshall Corporation. It assayed 18.1 per cent Pb, 25.4 per cent Zn, and 121.03 ounces Ag, and was a good ore for studying the effect of dichromate on the lead and zinc sulphides in particular.

The first test made of the Smuggler ore was with a charge of 500 grams stage-ground through 60-mesh in the dry condition.

No. 350 crude pinewood oil, Pensacola Tar & Turpentine Co., was used. This oil is especially effective for both bulk flotation and for flotation of the sphalerite in dichromate solutions.

Bulk flotation was practiced on the crude ore after which the overflow was given a roughing treatment in dichromate solution without any acid present. In treating the Mendota ore (p. 21) tests were made in which acid was absent or present in varying quantities. The dichromate was not consumed in this test, as shown by titrations before and after flotation.

Table 3 gives the results.

TABLE 3.—*Flotation of Smuggler ore and treatment of froth in dichromate solution without acid.*

Product.	Weight of product, grams.	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	500	100.0	19.5	100.0	26.2	100.0	6.4	100.0
Zinc product	165	33.0	9.0	15.2	48.7	61.2	5.9	30.4
Middling	89	17.8	15.5	14.1	42.6	29.0	5.7	15.9
Lead-iron underflow	150	50.0	42.5	65.5	6.6	7.6	10.1	47.3
Tailing	85	17.0	.8	.7	1.4	.9	1.2	3.2
Footing	489	97.8	.....	95.5	.....	98.7	.....	96.8

The feed analysis shows the ore to be very high in sulphides. If the "zinc product" and "middling" are combined, the zinc recovery is 90.2 per cent in a product assaying 11 per cent Pb, 46 per cent Zn, and 5.8 per cent Fe. The "lead-iron underflow" contains 65.5 per cent of lead in a product assaying 42.5 per cent Pb, 6.6 per cent Zn, and 10.1 per cent Fe. In connection with this product, the results show 47.3 per cent of the iron with the lead. These figures indicate a fair separation of the galena and pyrite from the sphalerite.

The ore was so obviously amenable to this process when perfected that more attention was given to further research than to directly obtaining good commercial results.

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Screen analysis of the "zinc product" showed that 97 per cent of the lead therein was minus 200-mesh. The outstanding feature indicated by this analysis is the segregation of the lead in the fines. The fine lead is the product difficult to remove from the zinc; the neutral dichromate solution does not retard it effectively, nor do the acid solutions. The iron is not segregated to any appreciable extent on the various meshes of the zinc product.

The conclusion drawn is that the fine lead will be a troublesome factor, whereas the iron, either fine or coarse, is more effectively acted on by the dichromate.

IRON CITY, AT BLACKHAWK.

The Iron City mill at Blackhawk, leased by the Midwest Mining & Milling Co., in its present practice makes a table lead-zinc-iron middling which is not salable because not capable, on account of the high content of pyrite, of being "graded up" enough. This product, assaying 10.6 per cent Pb, 11.4 per cent Zn, 27.2 per cent Fe, 0.40 per cent Cu, 0.48 ounce Au, and 7.48 ounces Ag, was studied, first, to observe the effect of the dichromate solutions, and, second, to eliminate iron and to recover some of the lead.

This middling, containing such a high proportion of sulphides, did not require a bulk-flotation treatment before being subjected to the dichromate solution.

A series of comparative runs with 500-gram charges was made to determine the amount of acid or alkali necessary in a dichromate solution to give the best separation of the three sulphides. The amount of dichromate was kept constant, but the acidity and alkalinity were changed. The following procedure was used: As the product coming from the mill contained material as coarse as 20-mesh, it was screened on 120-mesh; the coarse was ground wet in a porcelain jar, then the fines and some No. 350 Pensacola Tar & Turpentine Co. oil (crude pine oil) were added to the charge, and the grinding was repeated. The ground mixture was charged into a 4-liter machine with a solution containing 2 grams dichromate (0.05 per cent) and varying amounts of H_2SO_4 or Na_2CO_3, according to Figure 2. The condition of the solutions ranged, on the one hand, from neutral to an acidity of 1 c. c. concentrated H_2SO_4, and, on the other hand, to an alkalinity of 5 grams Na_2CO_3. The ordinates represent the recoveries of the various sulphides in the overflow; that is, a low extraction corresponds to a considerable retardation of the mineral from floating, and a high extraction indicates ready flotability.

An examination of the curve for lead shows that this sulphide floats very well (82 per cent) in the strongest alkaline solution em-

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ployed, and also that it floats moderately well (55 per cent) in the strongest acid solution used. The curve for lead, however, has a minimum point corresponding to 0.3 c. c. acid, in which lead is least flotable (26 per cent).

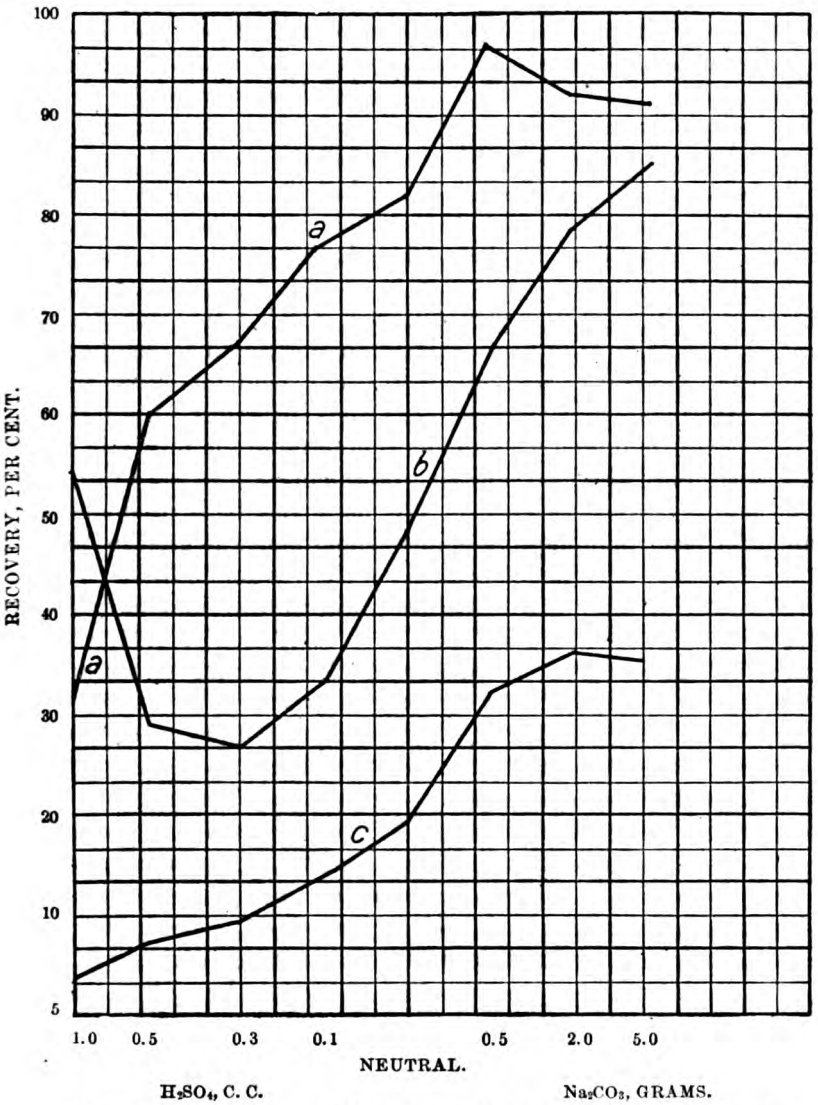

FIG. 2.—**Extraction of galena, sphalerite, and pyrite in a dichromate solution of 4 liters HNO_3 to 2 grams $Na_2Cr_2O_7$, with certain proportions of sulphuric acid and sodium carbonate.** *a*, Zinc; *b*, lead; *c*, iron.

The curve for iron indicates a progressively greater retardation as the solutions decrease in alkalinity and increase in acidity. However, it is known that a solution containing 4 c. c. or 5 c. c. acid would result in less retardation or in more iron in the overflow.

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The recovery of the zinc increases rapidly from the strongest acid solution to the solution containing 0.5 gram Na_2CO_3 (97 per cent), and then decreases slightly. The recovery of zinc in the acid solutions is low because of the large quantity of chemically combined iron. To determine the combined iron, a sample of the product as received from the mill was sized between 20-mesh and 30-mesh. With the aid of a microscope 1 or 2 grams of sphalerite was sorted free from the rest of the material. An analysis showed 7.6 per cent Fe in this sphalerite. For an ore free from chemically combined iron, the zinc-recovery curve would lie considerably above the one shown for this ore.

In the ores tested the types of sphalerite containing iron were more reluctant to float in the acid dichromate solution than those practically free from iron.

From a study of these curves it was decided that for the most effective separation, the best solution to use would be one containing a slight amount of acid, about 0.10 c. c. acid for every gram of sodium dichromate.

Tests of this middling were made to determine the probable grades and recoveries of the lead and zinc products.

The material was screened on 120-mesh and both products were panned for lead; panning in a gold-washing pan was used in place of tabling for small charges of ore. Then it was found best to grind the charge with the oil, which caused the zinc to float more freely. After grinding, the charge was floated in a solution of 0.05 per cent dichromate made up as stated above. No cleaning was practiced. However, a fair grade of zinc concentrate resulted, as evidenced by the assay of 5.4 per cent Pb, 41.6 per cent Zn, 12.4 per cent Fe, with a recovery of 90.5 per cent of the zinc. As about 7.6 per cent Fe was combined with the zinc, it is easily seen that the elimination of pyrite from the zinc was quite thorough. The lead concentrate assayed 56 per cent Pb, 2.3 per cent Zn, 13.5 per cent Fe, with a recovery of 66 per cent of the lead. The iron product contained 17.3 per cent of the lead, part of which might have been recovered; 81 per cent of the iron was thrown into the iron underflow. A test using a stronger dichromate solution (0.10 per cent) resulted in the zinc being "graded up" to 46.8 per cent. In a thorough treatment of this middling, the last part of the overflow would be diverted from going as a finished zinc concentrate and would be regarded as a middling to be reground and returned to the circuit for further treatment. Thus the grade of the concentrate would be improved.

In these tests, no attention was given to the recovery of the chalcopyrite and the precious metals, and, as a result, only a relatively small proportion of them was obtained with the lead. The copper, especially, and about 40 per cent of the silver segregated with the zinc

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product. Later in the work considerable attention was given to these metals in other ores, and fairly good recoveries resulted.

In this middling from the Iron City mill the pyrite carries practically no precious metals. The gold and silver are mainly in the galena and less in the chalcopyrite.

MENDOTA, AT SILVER PLUME.

Several samples from the Mendota mine at Silver Plume, operated by the Wasatch Colorado Mining Co., were brought to the laboratory.

This ore was the first studied in some detail and for which a fairly definite treatment of the lead, zinc, and iron sulphides was determined. Approximate figures on grades of products and recoveries were obtained by locked tests.

The first tests made were in continuation of the study of the action of the dichromate solution on these three sulphides. The general procedure was to practice bulk flotation on 500-gram charges of the ore ground through 80-mesh, discard the tailing, and then subject the rougher overflow to the action of the dichromate solution. No. 350 Pensacola Tar & Turpentine Co. oil was used.

When a rougher overflow was cleaned in ammoniacal dichromate solution (0.10 per cent)—it is, of course, understood that a base added to a solution of dichromate converts it into the normal chromate—there was seemingly no separation of lead and zinc. A voluminous froth appeared in this cleaning.

No separation of lead and zinc was observed in caustic soda dichromate solution; practically the entire rougher overflow floated. Again the conclusion was reached that the dichromate, to be effective in separating lead and zinc, must be either neutral or slightly acid—preferably acid. Excess acid must be avoided for reasons already mentioned, and for the reason that the dichromate is then very rapidly consumed and can not be used again.

The five tests following were made to obtain information as to the quantity of dichromate required and to compare presence and absence of acid. Table 4 indicates the results obtained with respect to the flotability of the three sulphides. In these tests the rougher overflow of the feed was separated by flotation in dichromate into only two products, namely, the zinc overflow and the lead-iron underflow.

TABLE 4.—*Relative floatability of galena, sphalerite and pyrite in specified amounts of dichromate and sulphuric acid (4-liter machine).*

Amount of reagent.		Zinc overflow, per cent of total of each metal in product.			Lead-iron underflow, per cent of total of each metal in product.		
$K_2Cr_2O_9$.	H_2SO_4 (c.c.)	Lead.	Zinc.	Iron.	Lead.	Zinc.	Iron.
4 grams	.....	39.3	81.3	30.5	49.0	12.5	56.4
4 grams	1.0	5.1	78.2	15.6	92.0	22.6	69.0
3 grams	1.0	70.9	90.1	34.3	27.5	3.6	51.0
2 grams	.1	87.8	92.8	67.5	12.9	2.3	18.5
1 gram	.5	90.7	94.0	59.6	7.9	4.6	31.9

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The feed assayed 11.5 per cent Pb, 11.3 per cent Zn, and 8.3 per cent Fe. The acid in the last run was cut down to 0.5 c.c., as 1 c.c. per 1 gram dichromate is an excess (0.73 c.c. acid is enough to release all the oxygen in 1 gram dichromate).

A comparison of the first two tests shows that neutral dichromate is effective in retarding the flotation of the lead and iron, but the effect is greater when some acid is added. Furthermore, a study of the last four tests shows that the lead and iron are kept in the underflow less effectively as the amount of dichromate is decreased. With a better understanding of this method a great economy of dichromate was obtained.

Neutral dichromate, as indicated by titration, was practically unconsumed, slightly acid dichromate was but slightly consumed, whereas strongly acid dichromate was almost instantaneously consumed. These tests, with other preliminary tests, were the basis for deciding to use slightly acid dichromate for the elimination of the iron and some of the lead from the zinc; the exact amount of dichromate used being dependent on the particular test. The ratio of acid to dichromate adopted was 0.10 c.c. to 0.15 c.c. acid per gram of dichromate.

These differences in the consumption of dichromate were further verified by tests with galena in neutral and acid solutions. The following is an illustration:

Pure galena (25 grams) ground through 60-mesh was shaken with 2 grams $K_2Cr_2O_7$ in 300 c.c. water. The mixture was filtered and titrated with thiosulphate. The test was repeated, using 0.05 c.c. sulphuric acid added to water, and again with 0.25 c.c. acid added to the water. Table 5 records the relative consumption of the dichromate.

TABLE 5.—*Relative consumption by galena of 2 grams dichromate in 300 c.c. water with stated amounts of sulphuric acid.*

Duration of shaking.	Neutral dichromate.	Dichromate plus 0.05 c.c. H_2SO_4.	Dichromate plus 0.25 c.c. acid.
0.0 minute	0.00	0.00	0.00
2 minutes	.07	.09	.89
6 minutes	.08	.09	.89
10 minutes	.08	.09	.88
25 minutes	.10	.11	.89

The dichromate is but slightly consumed in neutral or barely acid solution, but is partly consumed in a solution containing a small amount of acid.

Three flotation runs were made in the 9-liter machine with 2,000-gram charges to determine the grade and the recovery of products

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possible when the amount of dichromate was 4, 3, and 2 grams, respectively (⅛ c.c. acid per gram of dichromate).

The procedure was to dry grind the ore through 60-mesh and then wet grind with No. 2 Barrett oil (coal-tar cresote). When the charge was floated, the first part of the overflow was a lead product (the Barrett oil had been found to make a partial separation between lead and zinc). The second part obtained by the use of No. 350 Pensacola Tar & Turpentine Co. oil contained the remainder of the sulphides, leaving a clean tailing. The lead product being floated with the Barrett oil was panned for a finished lead concentrate; the overflow from the panning was combined with the remainder of the rougher overflow. This was given a roughing flotation in the dichromate solution, making a zinc concentrate overflow and a lead-iron underflow. The lead concentrates from panning averaged 77.0 per cent Pb, 4.2 per cent Zn, and 3.9 per cent Fe; the recovery was 73.84 per cent. The reason for the high recovery of lead, considering the crudeness of the method employed, was that the Barrett oil floated nearly all of the lead in advance of the zinc.

The grades of the "lead-iron underflow" and the "zinc concentrate" varied in the three roughing treatments in the dichromate solutions.

The recoveries of the various sulphides in the different products in the tests are indicated in Table 6.

TABLE 6.—*Percentage of the lead, zinc, and iron contained in the Mendota zinc concentrates and the lead-iron underflows for stated strength of acid dichromate.*

	Percentage of feed content.			
	Lead.	Zinc.	Iron.	
4 grams $K_2Cr_2O_7$+0.5 c.c. acid	12.63	83.41	22.72	Zinc concentrate.
3 grams $K_2Cr_2O_7$+0.375 c.c. acid	14.42	84.42	28.10	
2 grams $K_2Cr_2O_7$+0.25 c.c. acid	16.64	86.66	44.36	
4 grams $K_2Cr_2O_7$+0.5 c.c. acid	9.05	3.53	37.17	Lead-iron underflow.
3 grams $K_2Cr_2O_7$+0.375 c.c. acid	9.94	5.76	41.74	
2 grams $K_2Cr_2O_7$+0.25 c.c. acid	5.10	2.54	29.34	

The zinc recoveries increase slightly with decreasing amounts of dichromate, but the iron in the zinc increases rapidly with decreasing amounts. The test with 4 grams of dichromate (0.10 per cent solution) indicates the best results, especially if the grade of the zinc concentrate be considered—6.0 per cent Pb, 48.2 per cent Zn, and 8.5 per cent Fe; the zinc assay is several per cent higher than in the other two tests.

Screen analysis of the zinc product in the first run was made, to locate the loss in lead and to determine if fine lead predominated in the overflow from the dichromate treatment. The product was about 50 per cent plus 300-mesh (Multi-Metal sieve). The assay per cent of the coarse was 1.2 per cent Pb and of the fines was 9.6 per cent, showing a great proportion of fine lead with the fine zinc; 82 per cent

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of the lead in the zinc concentrate was minus 300-mesh. This indicates how serious a factor the fine lead is in grading up the zinc. The iron did not segregate.

Screen analysis was made of the lead-iron underflow to locate the zinc losses and to determine if coarse zinc predominated in this product. The sample was sized into four parts: Plus 100-mesh, minus 100 plus 150, minus 150 plus 200, and minus 200; 50 per cent was minus 200-mesh. From the chemical analyses it was easy to see that the zinc lost in the lead-iron underflow was coarse; 92 per cent of the zinc in this product was plus 150-mesh. The assay of the plus 100-mesh portion was 30.8 per cent Zn, of the minus 100 plus 150-mesh 6.8 per cent Zn, and of the minus 200-mesh material only 0.6 per cent Zn. These figures indicate the great segregation of the zinc in the coarser meshes. Of the lead in this product (the lead-iron underflow) about 50 per cent was coarser than 200-mesh. Again it was noted that the iron did not segregate appreciably on the various meshes.

These results present the problem of the separation of these three sulphides in a striking manner. The fine lead is the material contaminating the zinc concentrate. Slimed lead consequently must not be produced any more than is necessary in grinding, and as much as possible it should be removed before the dichromate treatment. On the other hand, the grinding must be fine enough to avoid considerable losses of zinc in the lead-iron product. The attempt has been made to remove as much lead as possible in the early stages of the treatment and then grind the zinc-iron product remaining through at least 100-mesh before floating in the acid dichromate solution.

The advantageous feature indicated by the analyses given above is that the iron does not segregate and may be removed from the zinc about equally well, whether it is coarse or fine; this accounts for the emphasis laid on the fact that the chief function of the dichromate is to eliminate the iron from the zinc. In most, if not all, of the Colorado ores examined the pyrite has been harder to crush than the other sulphides and almost as hard as the gangue material.

A screen analysis, Table 7, of a zinc concentrate and of the two zinc middlings made in another test, indicates the fineness of the product floating and the increasing coarseness of the middlings.

TABLE 7.—*Comparison of screen analyses of zinc concentrate and zinc middlings.*

Product.	Per cent zinc.
Zinc concentrate:	
+200-mesh	8
−200-mesh	92
Second zinc middling:	
+200-mesh	28
−200-mesh	72
First zinc middling:	
+200-mesh	44
−200-mesh	56

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To determine an approximate flow sheet several runs were made, different steps of the procedure being varied.

With the information gained from these tests a locked test of 12,000 grams of ore was made. The galena and sphalerite were readily floated, the extraction of each being 93 per cent. Advantage was taken of the rate of floatability of the lead with respect to the zinc by using No. 2 Barrett oil. The lead product was panned for a high-grade lead concentrate, and the tailing from the panning joined the zinc-iron froth which was made with No. 350 Pensacola Tar & Turpentine Co. oil. The rougher flotation tailings from the crude ore retained about one-half of the iron, but the dichromate solution was required to remove the iron remaining in the zinc-iron overflow. The recoveries of lead and zinc were each 80 per cent. The lead concentrate assayed 65 per cent Pb, 7 per cent Zn, and 7 per cent Fe. The zinc concentrate assayed 3 per cent Pb, 54 per cent Zn, and 6 per cent Fe.

The feed sample assays 7.80 ounces Ag, and the assays of several products show that the silver is carried by the lead. Further detail of the locked testing is omitted because the scheme was not satisfactorily developed at the time the work was done. The reagents used per ton of feed were 0.6 pound No. 2 Barrett oil, 1.2 pounds No. 350 Pensacola Tar & Turpentine Co. oil, 4.9 pounds dichromate, and 0.7 pound sulphuric acid.

GILPIN-EUREKA, AT CENTRAL CITY.

This table product came from the Gilpin-Eureka mill, at Central City, owned by the Gilpin-Eureka Mines Co.

A sample of a lead-zinc-iron middling not amenable to further concentration in the current practice of the mill was briefly examined to determine its physical condition and its behavior under laboratory methods of concentration. The sample assayed 8.8 per cent Pb, 8 per cent Zn, 19.3 per cent Fe, 0.86 per cent Cu, 0.66 ounce Au, and 4.26 ounces Ag.

In the part coarser than 120-mesh the galena content was low, but the part finer than 120-mesh contained enough galena to justify gravity concentration. With a gold-washing pan 52 per cent of the lead in the original sample was recovered. This lead concentrate assayed 52.6 per cent Pb, 3.6 per cent Zn, 16.4 per cent Fe, 1.62 ounces Au, and 10.70 ounces Ag. The percentage recovered and the grade of product are good evidence that an adquate mill would reduce waste of lead, gold, and silver in treating this type of ore.

The coarse material was given a brief grinding in a porcelain jar, after which the fine material and oil (No. 350 Pensacola Tar & Turpentine Co.) were added to it for further grinding.

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In the roughing flotation treatment of the feed sample, which obviously might have been avoided, about 95 per cent of the charge floated. The overflow was then given a roughing treatment in a 0.05 per cent dichromate solution (the ratio of dichromate to acid was 1 gram dichromate to 0.10 c.c. concentrated sulphuric acid), used at the rate of 4 pounds per ton of feed sample. In this step 71.4 per cent of the iron was eliminated in the underflow with an accompanying loss of only 5.69 per cent of all the zinc—a marked separation of minerals that formerly had floated together. The overflow was given a double cleaning in the used dichromate solution. Though about one-third of the zinc was withheld in the middling, 55 per cent of it entered a concentrate assaying 9 per cent Pb, 43.9 per cent Zn, 7.4 per cent Fe, and 4.80 per cent Cu. In another test a zinc concentrate assaying 47.8 per cent Zn was made. No effort was made to control the copper, gold, and silver, and they were badly segregated in the zinc concentrates. It is likely that by methods later developed a good percentage of these metals could be recovered with the lead. Elimination from the zinc of 93 per cent of the iron might be expected.

WILFLEY, AT KOKOMO.

The Wilfley mine, at Kokomo, is operated by the Elk Mountain Mining & Milling Co.

A sample of the zinc ore from this mine assayed 0.4 per cent Pb, 11.7 per cent Zn, 23.1 per cent Fe, and 0.88 ounce Ag. Its treatment involves the separation of the pyrite from the sphalerite in order to make a satisfactory zinc concentrate.

Stage grinding reduced the ore to 120-mesh, and flotation of the crude ore with No. 350 Pensacola Tar & Turpentine Co. oil eliminated 60 per cent of the feed content (70 per cent of the iron content). The zinc extraction was 95 per cent. The large percentage of pyrite discarded in the tailing is favorable. Possibly a satisfactory grade of zinc concentrate can be made by thorough cleaning in neutral water, but this was not tried. A single charge was cleaned in acid dichromate solution and the grade of zinc concentrate was 0.6 per cent Pb, 46.1 per cent Zn, and 13.2 per cent Fe. The recovery of 67 per cent of the zinc was too large—that is, for thorough cleaning more of the zinc should have been diverted to the middling circuit and a richer zinc concentrate left. A concentrate containing 50 per cent might be expected in a more complete test.

WHITECAP, AT LEADVILLE.

The Whitecap mine, at Leadville, is one of the Yak group and is operated by the American Smelting & Refining Co. Ore from this mine has a high pyritic content—50 to 60 per cent pyrite. The

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sample which was tested and given only a preliminary investigation assayed 3 per cent Pb, 11.6 per cent Zn, 25 per cent Fe, and 0.79 ounce Ag. No determined effort was made to recover the lead, though in one test gravity concentration recovered about one-third of it in a product very low in zinc.

Grinding to pass 120-mesh seems to be required. In a test where the grinding was somewhat coarser 41 per cent of the feed was discarded as a tailing that carried 52 per cent of the iron. The reluctance of the pyrite to float is advantageous. The lead and zinc extractions were 93 and 96 per cent, respectively.

Contrary to previous practice in former tests, No. 2 Barrett oil and No. 2 Cleveland Cliffs oil along with 2 pounds of sodium carbonate per ton of ore were used in the roughing treatment. The reason for grinding to 120-mesh is that the sphalerite coarser than this does not float well in the dichromate solution. A zinc concentrate assaying 4.5 per cent Pb, 50.4 per cent Zn, and 9.2 per cent Fe was made. It is likely that a concentrate closely approaching this grade could be maintained by using 2 pounds of acid sodium dichromate per ton of mill feed. The zinc recovery should be 85 per cent. These conclusions are based on the results of several separate tests. No locked testing was done to clean the middlings.

JESSIE, AT BRECKENRIDGE.

The Jessie mine and mill, at Breckenridge, are owned and operated by G. F. Roth. The sample from the mill was a zinc-iron middling—the table discharge between the lead concentrate and the siliceous tailing. Its pyrite content was high—about 60 per cent. This sample was of peculiar interest because a part of the zinc was combined with the iron in the form of marmatite, a magnetic sphalerite. The object of the investigation was to test a process with this type of ore rather than to test the ore. A résumé of the results of a locked test appears in Table 8:

TABLE 8.—*Résumé of a locked test of middling from the Jessie mine.*

Product.	Weight of products (grms).	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	10,000	100.00	1.00	100.00	17.70	100.00	31.40	100.00
Zinc concentrate	3,062	30.62	2.26	69.11	43.08	74.52	15.14	14.70
First middling	180	1.80	1.20	2.16	7.60	.77	37.70	2.16
Second middling	140	1.40	1.10	1.59	10.40	.82	35.80	1.58
Third middling	207	2.07	1.00	2.07	8.30	.97	35.90	2.37
Fourth middling	335	3.35	1.30	4.35	12.60	2.38	32.00	3.41
Iron underflow	1,227	12.27	1.15	14.15	2.02	1.39	42.26	16.51
Tailing	4,855	48.55	.25	12.08	7.72	21.17	37.90	58.56
Footing	10,006	100.06	.....	105.51	.....	102.02	.....	99.29

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The feed sample had to be ground in the presence of the oil to insure flotation of the sphalerite, probably because the sample was shipped in the form of wet pulp. The mesh was minus 120. The roughing flotation of the feed was in five 2,000-gram runs in the 9-liter hollow-shaft machine. The five mixed-sulphide overflows, each of which weighed about 1,000 grams (dry weight), were treated successively in locked-testing fashion in the 4-liter hollow-shaft machine. The reagents required per ton of feed were 1.5 pounds No. 350 Pensacola Tar & Turpentine Co. oil, 4 pounds sodium bicarbonate (in the roughing flotation of the feed), and 2 pounds acid sodium dichromate. In the roughing treatment of the mixed-sulphide overflows 1 gram of the dichromate was used, and in the four subsequent cleanings 0.5 gram was added to the first two.

Four cleanings after the roughing flotation in dichromate were regarded as necessary to make the results comparable with the work of a six-cell commercial machine. The first and second middlings were reground and with the third and fourth middlings joined the charge for the subsequent run. Of course, the charge had to be withdrawn to remove each middling, and the solution was decanted to enter the machine for a further cleaning of the corresponding overflow. At the end of each of the five runs the solution was thrown away.

The tailing (see Table 8) contained too much zinc (21.17 per cent of the zinc in the feed) to be satisfactory. Another test showed that a tailing of low zinc content could be made but a high loss appeared in the iron underflow. The probable remedy is obvious—the iron underflow could be reground and floated again for a further recovery of zinc.

As the table shows, 74.52 per cent of the zinc was recovered and 1.39 per cent was lost in the iron underflow; that is, 97 per cent of the zinc entering the dichromate treatment would be recovered in continuous operation.

The four middlings, remaining at the end of the test, contained 4.94 per cent of the zinc. Continuous running would recover 97 per cent of this, or $(\frac{97}{100} \times 4.95 = 4.7)$ 4.7 per cent additional zinc, increasing the total recovery to about 79 per cent of the zinc.

In another test with a 1,000-gram charge, a zinc concentrate made from one roughing and one cleaning in dichromate assayed 47.8 per cent Zn and 12.5 per cent iron. In another similar test the zinc concentrate assayed 48.8 per cent Zn and 12 per cent iron. It is likely that the density of the pulp in the locked test was too great; including middlings, the dry weight of the charge was about 1,500 grams—entirely too much for cleaning in a 4-liter machine.

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The copper, though in very small amount, was segregated in the zinc concentrate, and the ratio of lead to silver in the several products indicates that the silver was carried by the lead.

WELLINGTON, AT BRECKENRIDGE.

Through the courtesy of the Wellington Mines Co., of Breckenridge, a sample of the jig and table tailings from the mill was obtained for examination. Flotation samples were prepared by screening the mixture on 120-mesh and rejecting the coarse, the idea being that the original mixture, after grinding, could be concentrated if the fines containing the segregated clay proved to be amenable.

The sample of the natural minus 120-mesh material assayed 9.3 per cent zinc. The lead content was so low that it was not taken into account. The extraction obtained was 92.2 per cent of the zinc, the procedure being ordinary roughing and cleaning flotation in a locked test so that the underflow from the cleaner could be sent back to the rougher. Sodium bicarbonate (8 pounds per ton) was required with No. 350 Pensacola Tar & Turpentine Co. oil. Without the bicarbonate the flotation was not good.

Only 40 per cent of the iron was extracted with 92.2 per cent of the zinc. By treatment in acid dichromate a zinc concentrate assaying 42.2 per cent zinc and 7 per cent iron was made. The cleaning of the feed sample was not thorough enough for satisfactory removal of the clay-like gangue, which accounts for the low grade of the concentrate.

The chief object of the work was to show that the combination of sodium bicarbonate and oil, used so successfully in the roughing treatment of other ores, would apply to this material.

AKRON, AT WHITEPINE.

The Akron mines, at Whitepine, are operated by the Dorr Co. Several samples of the ore brought to the laboratory assayed about 15 per cent Pb, 16 per cent Zn, 11 per cent Fe, and 6 ounces Ag. As the Akron mines supplied a metallurgist, Mr. A. E. Sargent, to cooperate in the work, more attention than otherwise would have been possible was given to the Akron ore.

The company has a small mill which is equipped for table concentration, flotation, and magnetic separation, but at the time this work was undertaken the mill practice was not wholly satisfactory. Figure 4 is a flow sheet of the mill. The ore has a high content of argentiferous galena, sphalerite, and pyrite; some of its characteristics are as follows: (1) The ore is high-grade; (2) all the silver is combined with the galena; (3) crushing to 120-mesh is required to entirely unlock the economic minerals; (4) the gangue is decomposed, giving

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rise to much clay or what is commonly termed colloidal material; (5) soluble salts, some of them bicarbonates, are present which have a marked coagulating effect; (6) some of the lead is oxidized; and (7) limestone is abundant and is likely to contaminate the zinc concentrate.

The first flotation tests were surprising, as the extraction was only 73 per cent of the lead and 47 per cent of the zinc. Use of sodium bicarbonate to nullify the coagulating effect of the natural salts increased the extraction to more than 90 per cent.

In the first extraction tests the possibility of colloidal matter making trouble prompted a separate treatment of sand and slime, but this procedure was too complicated and later was rejected. The system finally adopted was to stage grind in jaw crusher and rolls through 40-mesh, and table for a coarse lead concentrate that part remaining on 120-mesh.

TABLE 9.—*Details of locked test of Akron ore.*

Product.	Weight of products (gms.).	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	10,000	100.00	15.6	100.00	16.3	100.00	10.8	100.00
Coarse lead concentrate	544	5.44	58.6	20.44	3.0	1.00	12.3	6.19
Flotation lead concentrate	557	5.57	43.6	15.57	24.3	8.31	3.8	1.96
Middling lead concentrate	26	.26	49.0	.82	1.3	.02	17.1	.42
Slime tailing	1,590	15.90	5.6	5.74	5.6	5.44	6.6	10.09
Sand tailing	3,020	30.20	1.6	3.09	2.0	3.71	15.7	43.90
Second coarse lead concentrate	110	1.10	28.1	1.98	.8	.05	29.0	2.95
Bulk-flotation lead concentrate	710	7.10	68.0	30.95	3.0	1.31	6.8	4.47
Lead concentrate from zinc concentrate	30	.30	60.0	1.16	4.2	.08	10.2	.29
Zinc concentrate	2,637	26.37	7.3	12.34	47.8	77.65	5.8	14.16
First middling	112	1.12	14.0	1.00	8.2	.57	23.5	2.44
Second middling	107	1.07	13.8	.95	11.8	.78	23.2	2.30
Third middling	72	.72	15.8	.73	16.3	.72	10.5	.70
Iron underflow	349	3.49	11.3	2.52	6.3	1.36	30.0	9.71
Footing	9,864	98.64	.....	97.29	.....	101.00	.....	99.58

A high-grade lead concentrate containing 20.44 per cent of the lead was obtained (see Table 9). The table tailing was ground wet in the jar to pass 120-mesh and combined with the other fines for bulk flotation. It was decided to recover the finest lead by flotation before bulk-floating, because this lead was the most difficult to recover on the table, and a 3:1 mixture of coal tar and cresylic acid was used. The resulting product, flotation lead concentrate, was too high (24.3 per cent) in zinc and exposed the weakest step in the treatment. In bulk flotation No. 350 Pensacola Tar & Turpentine Co. oil and sodium bicarbonate were used. The bulk overflow was returned for cleaning, chiefly in order to eliminate the muddy water. The granular middling resulting from the cleaning was tabled to

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recover a middling lead concentrate, and the table tailing was reground and refloated, the tailing from this reflotation being added to the other tailings. In mill practice the middling would have been tabled (if feasible), reground, and returned to the rougher, but this procedure was not easy in the laboratory.

The flotation tailing was separated into slime tailing and sand tailing by rough decantation. The sand tailing was tabled for second coarse lead concentrate, which added 1.98 per cent to the lead recovery.

Lead to the amount of 5.6 per cent in the slime tailing seems high, but the fact that this tailing assays 2.3 per cent in oxidized lead, makes it appear in a more favorable light; a good slime table could have removed some of this lead, but its trial in the laboratory was not feasible.

The mixed sulphide from bulk flotation was tabled for bulk-flotation lead concentrate and thus 30.95 per cent of the lead, in a rich product, was added to the lead concentrate. At this point 69.76 per cent of the lead had been recovered, and the zinc-iron concentrate left for cleaning had a calculated assay value of 8.8 per cent Pb, 40 per cent Zn, and 9.7 per cent Fe, representing 81.16 per cent of the zinc.

This zinc-iron product was treated by flotation in an acid dichromate solution, where it received one roughing and three cleanings. The zinc overflow obtained was passed over the table, not because such procedure seemed feasible in practice but to determine the behavior of the lead it contained. A small amount of a 60 per cent lead concentrate was made as is shown by lead concentrate from zinc concentrate, Table 9. The grade of the zinc concentrate, shown in Table 10, did not come up to expectations. In a former locked test of six runs, the assays of the zinc concentrates were 51.8, 53.5, 53.8, 50.0, 53.5, and 54.7 per cent zinc. This will be referred to again.

The grades of the first, second, and third middlings show the retardation of the lead and iron in flotation.

The iron underflow, with its 11.3 units of lead, low zinc content, about 60 per cent pyrite, and 4.27 ounces silver, was regarded as of marketable value and was added to the lead concentrate. Obviously the lead and silver in this product would have gone into the zinc concentrate if roasting and magnetic separation had been practiced without subsequent tabling.

A résumé of the results is given in Table 10. The absolute recovery of the lead was 73.42 per cent. Subsequent treatment in a fresh solution would have made nonfloatable nearly all of the lead in the zinc middling, which would have been recovered in the iron under-

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flow. Correction of the error in "footing" also would add to the lead recovery, so that with these two corrections a recovery of 78 per cent of the lead is indicated. The oxidized lead in the feed sample amounts to 1.2 per cent—that is, 7.7 per cent of the lead in the feed is oxidized.

In a comparison of the dichromate treatment with a possible magnetic separation for grading up the zinc, it is important to note that only 1.36 per cent of the zinc in the feed was lost in the iron underflow.

TABLE 10.—*Résumé of locked test of Akron ore.*

Product.	Weight of products (grams).	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	10,000	100.00	15.6	100.00	16.3	100.00	10.8	100.00
Lead concentrate	2,326	23.26	49.3	73.42	8.5	12.12	12.1	25.98
Zinc concentrate	2,637	26.37	7.3	12.34	47.8	77.65	5.8	14.16
Zinc middling	291	2.91	14.3	2.68	11.6	2.07	20.2	5.44
Tailing	4,610	46.10	3.0	8.87	3.2	9.17	12.6	53.62

A return may now be made to a consideration of the zinc concentrates referred to above. In the former locked test (the one that gave the higher grade of zinc concentrate) only a part of the zinc received the treatment in dichromate; the first part of the rougher zinc froth already low in iron was given a cleaning in a neutral water circuit. The froth from this cleaning gave a final zinc concentrate and the underflow joined the last part of the rougher zinc froth to be given a cleaning in the dichromate circuit. This procedure reduced the bulk of material to be treated in the dichromate circuit and thus made possible a more dilute pulp. The most desirable pulp density has not been accurately determined, but it seems that it should not exceed 6 to 1. In the former test where the high-grade zinc was made, the percentage of zinc recovered was not quite so high as in the test being described, but the fault was in the method of recovery of the lead, that is, the zinc content of the lead was too high. In grading up zinc concentrates the above deportment should constantly be kept in mind, that is, the first part of the zinc froth is relatively of low iron content (when no chemicals are used), whereas near the end of the run when an effort is being made to obtain tailings of low zinc content, much iron is likely to float and join the zinc in the froth. Such a segregation, so simply obtained, should at all times be used to advantage.

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The dichromate consumption was 2 pounds per ton of feed, oil 1 pound, and sodium bicarbonate, in the roughing of the feed, 6 pounds.

The lime content of the zinc concentrate was kept below 0.3 per cent.

RICO-ARGENTINE, AT RICO.

The Rico-Argentine mine, at Rico, is owned and operated by the Rico-Argentine Mining Co. The sample brought to the laboratory was taken from the dump of the Argentine tunnel. This sample, though high grade as a milling ore, was waste rock in the operations at Rico, being too-low grade for shipment to the oxide plant. Its analysis appears in Table 11.

The absence of milling at Rico aroused suspicion as to the amenability of the ore. A microscopic examination showed the degree of grinding required to unlock the sulphides was 80-mesh. After a preliminary flotation test a locked test was made, the mill feed being tabled for a lead concentrate (the recovery was 56.64 per cent of the lead); the table tailing being subjected to bulk flotation, which gave a high extraction but left only 18.17 per cent of the feed in the tailing.

TABLE 11.—*Résumé of locked test of Rico-Argentine ore.*

Product.	Weight of products (grms.).	Per cent of total weight.	Lead.		Zinc.		Iron.		Copper.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	3,000	100.00	14.8	100.00	22.0	100.00	10.7	100.00	1.00	100.00
Lead concentrate	462	15.40	63.1	65.64	3.9	2.77	9.3	13.30	.09	1.34
Zinc concentrate	1,012	33.73	9.3	21.13	49.5	75.86	3.7	11.82	2.40	81.06
Middling	305	10.17	7.8	5.41	19.6	9.04	10.6	10.12	.77	7.86
Iron underflow	625	20.83	3.2	4.51	7.8	7.39	21.2	41.28	.32	6.67
Tailing	545	18.17	1.7	2.14	2.6	2.17	10.8	18.40	.09	1.78
Footing	2,949	98.30	.....	98.83	.....	97.23	.....	94.92	.....	98.71

The mixed sulphide from flotation was again tabled and 9 per cent was added to the lead recovery, making a total of 65.64 per cent. The zinc-iron tailing made on the table had the following calculated assay: 7.1 per cent Pb, 31.3 per cent Zn, 10.5 per cent Fe, and 1.5 per cent Cu. This was divided, without drying, into four charges of approximately 500 grams each. These charges constituted the material for four runs in a locked test of one roughing and three cleanings of each charge in acid dichromate. The middlings probably brought the successive roughing charges up to 800 grams, which was very heavy for the 4-liter machine.

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Because of the possibility that some of the coarse sphalerite might not be floatable in the acid dichromate solution, it was deemed best to grind the coarse material before beginning the flotation treatment. The zinc concentrate, as shown by Table 11, assays 49.5 per cent zinc. The iron underflow (Table 11) contains 41.28 per cent of the iron of the mill feed, showing that the dichromate was effective in removing iron from the zinc; the zinc concentrates assayed only 3.7 per cent Fe. The iron content of the zinc concentrate is very low for an ore containing so much of the sphalerite in the form of "black jack."

Bicarbonate, 3 pounds per ton, was required for roughing the mill feed with No. 350 Pensacola Tar & Turpentine Co. oil. The dichromate used amounted to 8 pounds per ton of mill feed. The large amount of dichromate employed requires explanation. It is partly justified by the high sphalerite content, nearly one-third of the mill feed. Moreover, the following modification of the treatment would allow great economy of the reagent without lowering the grade of concentrate. The first part of the zinc froth has a low iron content and can be diverted for a cleaning without chemicals. The tailing from this cleaning can be added to the rest of the zinc froth to enter the treatment in dichromate. Thus the tonnage of sulphides to be treated with chemicals is reduced. Further, this part should be given a brief grinding to make the sphalerite more floatable; the iron underflow contains 7.39 per cent of the zinc in the mill feed, this being higher than is generally expected.

The zinc concentrate (Table 11) assays 9.3 per cent Pb., and contains 21.13 per cent of the lead in the mill feed. This proportion is too high but indicates as good a separation as can be expected from a laboratory table. This loss is, of course, in excessively fine lead, enough of which should be recovered by a good slime table to bring the recovery up to 75 per cent.

The middling contains enough zinc to bring the expected zinc recovery up to 80 per cent.

The lead concentrate assays 0.30 ounces Au. The feed sample assays 4.36 ounces Ag. The lead concentrate contains 52 per cent of the silver with 65.64 per cent of the lead. The distribution of the silver throughout the several products, some of which are not shown in the table, indicates that more than three-fourths of it is combined with the lead.

The treatment was intentionally elaborate because at the beginning the ore was thought to be refractory. It proved a very simple ore to treat.

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HIDDEN TREASURE, AT SNEFFLES (OURAY).

The Hidden Treasure mine (at Sneffles, near Ouray) is owned by the Thomas F. Walsh estate. The ore was the first tested that contained an appreciable proportion of copper (chalcopyrite). The sample assayed well in lead, zinc, and silver, as shown by the assay of "feed," Table 12. The procedure of the treatment applied, as shown in Figure 2, shows the procedure of treatment.

By tabling that part of the feed coarser than 120-mesh, lead concentrate (1) was obtained, which as Table 12 shows is high grade and contains 39 per cent of the lead in the mill feed. The feasibility of this tabling could not be questioned, even though flotation were 100 perfect; the coarseness of this lead as compared with "flotation lead" is in itself sufficient justification for the table work required.

The reasons for recovering lead concentrates 2 and 5 (Table 12 and Fig. 3) by flotation were three. First, the copper content (chalcopyrite) though small, must be recovered by flotation to keep it from contaminating the zinc concentrate; second, the very fine lead can best be recovered by flotation; and, third, some of the silver is not amenable to gravity concentration and must be recovered by flotation. Products 2 and 5 contained 38.91 per cent of all the silver.

The cleaning of the zinc product in the acid dichromate solution eliminated 54 per cent of its iron content and lost only 7 per cent of the zinc in the mill feed.

The removal of the iron from the zinc concentrates was probably not as complete as it would have been if sodium sulphide had not been used in the first steps of the process, because sodium sulphide and dichromate have antagonistic effects on pyrite and must be considered.

Pyrite, when in a neutral flotation charge, is generally the least floatable of all the mineral sulphides; in the presence of sodium sulphide it becomes the most floatable, as is evidenced by the high iron content of concentrates 2 and 5—20.8 per cent and 27.7 per cent, respectively. Some of the iron is removed that would otherwise enter the dichromate solution with the zinc, but, on the other hand, the pyrite which does enter this circuit, having been sulphidized, must be acted on more vigorously by the acid dichromate to prevent it from floating in the zinc froth. As a result, more dichromate than otherwise would be required must be used.

Table 13 gives a résumé of the results of the locked test. The consumption of sodium sulphide and dichromate per ton of mill feed was 4 pounds and 1.6 pounds, respectively. The amount of sodium sulphide was excessive because of the unavoidably dilute pulp used in cleaning the second overflow; the machine was too large.

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TABLE 12.—*Details of locked test of Hidden Treasure ore.*

Product.	Weight of products (grams).	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	10,000	100.00	6.83	100.00	6.37	100.00	5.2	100.00
1. Lead concentrate	447	4.47	59.6	39.00	1.5	1.05	12.4	10.66
2. Lead concentrate	1,082	10.82	20.5	32.52	8.4	14.27	20.8	43.22
3. Lead concentrate	134	1.34	58.4	11.45	6.0	1.26	11.1	2.89
4. Lead concentrate	94	.94	52.8	7.26	3.2	.47	15.4	2.78
5. Lead concentrate	100	1.00	12.4	1.81	12.4	1.90	27.7	5.33
6. Iron underflow	520	5.20	2.0	1.52	4.0	3.26	6.8	6.80
7. Zinc concentrate	885	8.85	2.4	3.11	49.1	68.21	7.4	12.59
8. Zinc middling	302	3.02	3.2	1.41	14.0	6.63	13.8	8.01
9. Tailing	6,300	63.00	.2	1.84	.3	2.96	.4	4.85
Footing	9,864	98.64	.....	99.92	.....	100.01	.....	97.13

Product.	Weight of products (grams).	Per cent of total weight.	Copper.		Gold.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	10,000	100.00	0.44	100.00	0.02	100.00	10.92	100.00
1. Lead concentrate	447	4.47	.34	3.45	.11	24.58	53.81	22.03
2. Lead concentrate	1,082	10.82	2.22	54.54	.05	28.24	36.24	35.91
3. Lead concentrate	134	1.34	.38	1.15	.15	10.05	80.63	9.90
4. Lead concentrate	94	.94	.22	.47	.23	10.81	81.53	7.08
5. Lead concentrate	100	1.00	2.96	6.73	.05	2.50	32.87	3.00
6. Iron underflow	520	5.20	.16	1.89	.01	2.60	4.25	2.02
7. Zinc concentrate	885	8.85	1.20	24.10	.04	17.70	18.68	15.14
8. Zinc middling	302	3.02	.34	2.33	.02	3.02	11.28	3.12
9. Tailing	6,300	63.00	.04	5.73	.....	.....	.38	2.20
Footing	9,864	98.64	.....	100.39	.....	99.50	.....	100.40

TABLE 13.—*Résumé of locked test of Hidden Treasure ore.*

Product.	Weight of products (grams).	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	10,000	100.00	6.83	100.00	6.37	100.00	5.2	100.00
Lead concentrate	1,857	18.57	33.9	92.00	6.51	19.00	18.2	64.88
Zinc concentrate	885	8.85	2.4	3.16	49.1	68.20	7.4	12.59
Zinc middling	302	3.02	3.2	1.40	14.0	6.64	13.8	8.01
Tailing	6,820	68.20	.33	3.34	.58	6.24	.88	11.65

Product.	Weight of products (grams).	Per cent of total weight.	Copper.		Gold.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	10,000	100.00	0.44	100.00	0.02	100.00	10.92	100.00
Lead concentrate	1,857	18.57	1.57	66.34	.08	76.18	45.78	77.70
Zinc concentrate	885	8.85	1.20	24.10	.04	17.70	18.68	15.10
Zinc middling	302	3.02	.34	2.33	.02	3.02	11.28	3.12
Tailing	6,820	68.20	.05	7.62	.....	2.60	.68	4.22

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The following summary from Table 12 and Figure 3 shows the effectiveness of the reagents used in the roughing of the mill feed to float the lead, copper, gold, and silver in advance of the zinc. Products 2, 3, and 5 contain 45.78 per cent of the lead, 62.42 per cent of

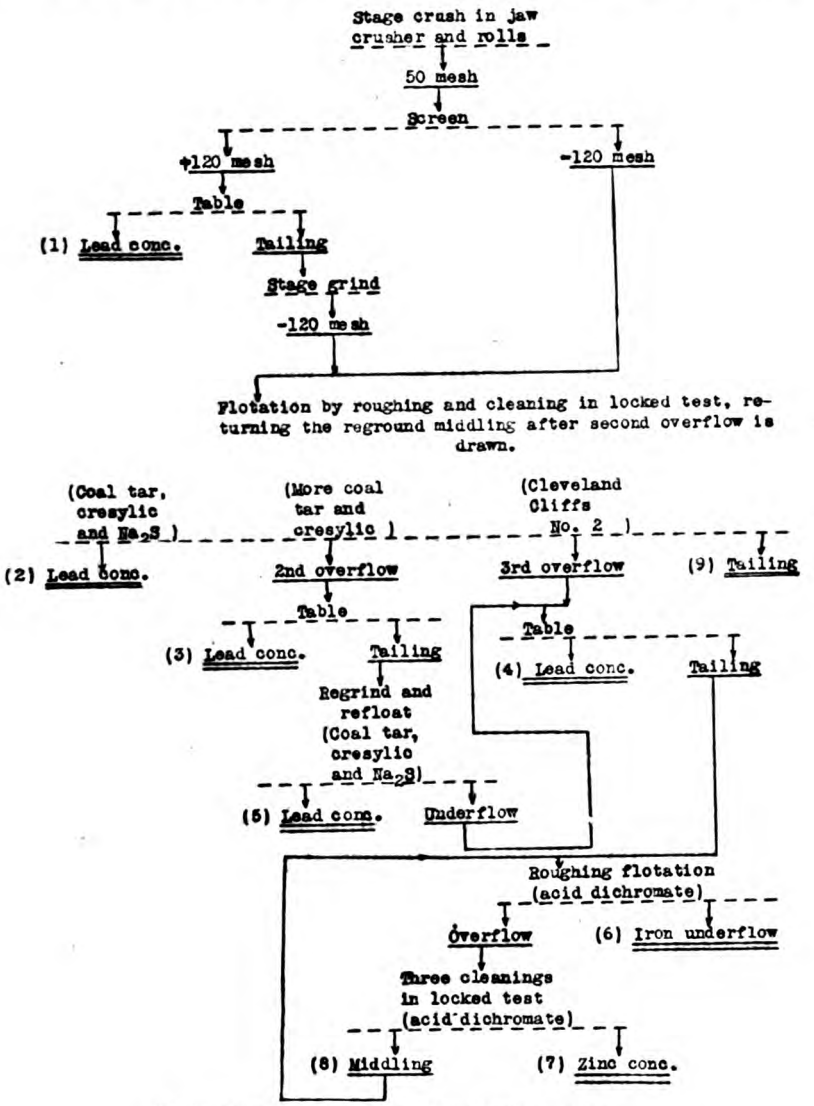

FIG 3.—Flow sheet of locked test of Hidden Treasure ore.

the copper, 40.79 per cent of the gold, and 48.81 per cent of the silver, with only 17.44 per cent of the zinc, and these recoveries were made after more than one-third of the lead and a goodly percentage of the copper and precious metals had been recovered in lead concentrate 1.

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Cleveland Cliffs No. 2 oil was used to finish the flotation, and a very low-grade rougher tailing resulted.

The lead recovery was 92 per cent, and the zinc in the zinc middling would bring the expected zinc recovery up to 73 per cent.

The idea of using coal tar and cresylic acid was borrowed from the Sunnyside mill practice. The coal tar (Denver Gas & Electric Co.) and cresylic acid were mixed in the ratio of 3 to 1.

IOWA-TIGER, AT SILVERTON.

The Iowa-Tiger mine, at Silverton, is owned and operated by the Iowa-Tiger Mining Co. The sample from the Iowa-Tiger mill was a tailing from the jigs making lead concentrates and zinc concentrates and in consequence contained the most refractory metalliferous minerals in the ore.

The sample was low grade, as shown by "feed," Table 14. The procedure followed was the same as for the Hidden Treasure ore (Fig. 3), except that the plus 120-mesh material was tabled to make a sand tailing which eliminated 31 per cent of the feed with allowable losses, and decreased the amount of grinding required.

Three comments may be made on Table 14 as follows: (1) The zinc concentrate was contaminated with about 16 per cent chalcopyrite. (2) Comparison of lead concentrates 2a and 2b shows that 2b is of better grade; two drops of the tar mixture was used, in the 4-liter machine, to draw 2b, whereas only one drop was used to draw 2a. This suggests that the nature of the flotation products responds to the reagents so readily that the products could be improved by further study—more of the copper could be kept out of the zinc concentrate. (3) The one hundred-fold increase of gold in 4 is proof that gold was carried in the froth made by the Cleveland Cliffs oil and was recovered by gravity concentration on the Wilfley table. Such a concentrate demands attention; it assays 7.14 ounces gold and 30.70 ounces silver, whereas the feed assays only 0.07 ounce gold and 1.07 ounces silver.

Table 15 shows that 74.88 per cent of the lead was recovered and the zinc content was kept far below the point of penalty. The zinc in the zinc middling brings the expected zinc recovery up to 70 per cent. The grade should be improved.

The chemicals required amounted to 3.5 pounds sodium sulphide and 0.9 pounds sodium dichromate per ton of mill feed. The amount of sodium sulphide was excessive because of the pulp being unnecessarily dilute when the second grab was cleaned. The plan of taking a second grab for further cleaning, as shown by Figure 3, had to be abandoned in subsequent work because of the difficulty in the limited amount of time of determining experimentally the most desirable

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quantity of reagents and the conditions. Doubtless the plan would prove effective in a mill.

TABLE 14.—*Details of locked test of Iowa-Tiger ore.*

Product.	Weight of products (grams).	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	17,000	100.00	1.75	100.00	2.6	100.00	2 45	100.00
1. Lead concentrate	185	1.09	41.2	25.62	3.8	1.59	20.0	8.88
2a. Lead concentrate	193	1.13	6.8	4.48	7.3	3.19	29.7	13.76
2b. Lead concentrate	356	2.09	8.6	10.20	6.0	4.83	26.2	22.39
3. Lead concentrate	131	.77	62.6	27.57	3.4	1.01	9.5	2.99
4. Lead concentrate	34	.20	42.8	4.89	14.7	1.13	13.2	1.08
5. Lead concentrate	90	.53	6.7	2.03	23.6	4.81	20.7	4.47
6. Iron underflow	470	2.77	3.2	5.06	4.6	4.89	4.7	5.30
7. Zinc concentrate	674	3.97	3.3	7.47	43.0	65.57	8.9	14.40
8. Zinc middling	165	.97	3.6	1.99	12.2	4.56	8.8	3.48
Sand tailing	5,270	31.00	.35	6.20	.4	4.77	.9	11.38
9. Tailing	9,200	54.12	0.15	4.64	0.2	4.21	.5	11.07
Footing	16,768	98.64	.....	100.24	.....	100.56	.....	99.20

Product.	Weight of products (grams).	Per cent of total weight.	Copper.		Gold.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	17,000	100.00	0.56	100.00	0.07	100.00	1.07	100.00
1. Lead concentrate	185	1.09	1.00	1.94	.83	12.90	8.43	8.58
2a. Lead concentrate	193	1.13	4.88	9.89	.28	4.54	4.48	4.76
2b. Lead concentrate	356	2.09	5.12	19.15	.30	8.88	5.06	9.90
3. Lead concentrate	1.31	.77	1.20	1.65	2.16	23.78	11.92	8.59
4. Lead concentrate	34	.20	1.68	.60	7.14	20.40	30.70	5.74
5. Lead concentrate	90	.53	8.92	8.43	.07	.53	4.87	2.41
6. Iron underflow	470	2.77	1.00	4.94	.12	4.74	3.92	10.13
7. Zinc concentrate	674	3.97	5.76	40.78	.26	14.73	5.94	22.01
8. Zinc middling	165	.97	2.56	13.25	.14	1.94	6.18	5.60
Sand tailing	5,270	31.00	.08	4.45	.01	4.43	.36	10.43
9. Tailing	9,200	54.12	.04	3.86	.005	3.86	.20	10.12
Footing	16,768	98.64	.....	100.13	.....	100.73	.....	98.27

TABLE 15.—*Résumé of locked test of Iowa-Tiger ore.*

Product.	Weight of products (grams).	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	17,000	100.00	1.75	100.00	2.60	100.00	2.45	100.00
Lead concentrate	989	5.82	22.52	74.88	7.40	16.56	22.56	53.57
Zinc concentrate	674	3.97	3.30	7.47	43.00	65.57	8.90	14.40
Zinc middling	165	.97	3.60	1.99	12.20	4.56	8.80	3.48
Tailing	14,940	87.88	.32	15.90	.41	13.87	.78	27.75

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TABLE 15.—*Résumé of locked test of Iowa-Tiger ore*—Continued.

Product.	Weight of products (grams).	Per cent of total weight.	Copper.		Gold.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	17,000	100.00	0.56	100.00	.07	100.00	1.07	100.00
Lead concentrate	989	5.82	4.01	41.66	.86	71.03	7.35	40.00
Zinc concentrate	674	3.97	5.76	40.78	.26	14.73	5.94	22.01
Zinc middling	165	.97	2.56	4.44	.14	1.94	6.18	5.60
Tailing	14,940	87.88	.09	13.25	.01	13.03	.37	30.65

COLUMBUS, AT SILVERTON.

The Columbus mine and mill, at Silverton, are operated by the Gnome Mining Co. As the metal content (see Table 16) of the Columbus ore is low, a flow sheet of maximum simplicity is imperative; the second overflow, as shown in the procedure of the Hidden Treasure (Fig. 3, p. 37), was omitted. A sand tailing was made to reduce to a minimum the amount of grinding. The procedure is shown in Figure 4. The details of the locked test are shown in Table 16, and the résumé in Table 17.

Although the lead concentrate was low grade, its zinc content was below the penalty point; it contained "pay copper," but the "insoluble" was about 7 per cent in excess of the iron. The lead recovery was 72.71 per cent, and the lead concentrate contained 94.60 per cent of the gold and 58.48 per cent of the silver.

The zinc concentrate assayed 52.0 per cent zinc and the zinc in the zinc middling brings the expected recovery of zinc up to 68 per cent.

The consumption of sodium sulphide and dichromate amounted to 1.3 pounds and 0.8 pound, respectively, per ton of mill feed.

The treatment in the acid dichromate solution eliminated 75 per cent of the iron (not as chalcopyrite) that entered the treatment and sacrificed only 4.54 per cent of the zinc in the mill feed.

The iron underflow produced by the dichromate treatment can be concentrated on tables when the situation justifies, or when rich enough in lead and precious metals it can be added to the lead concentrate.

A noteworthy feature of Table 16 is the high segregation of precious metals in concentrates 2 and 3.

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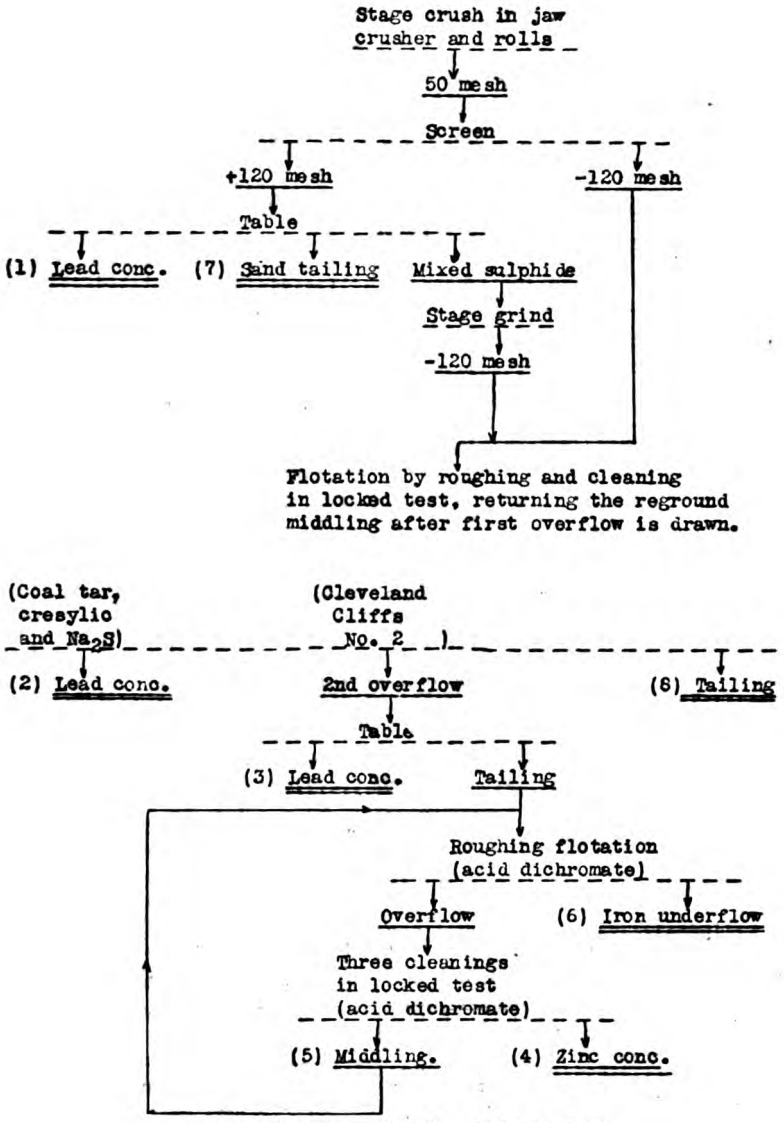

FIG. 4.—Flow sheet of locked test of Columbus ore.

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TABLE 16.—*Details of locked test of Columbus ore.*

Product.	Weight of products (grams)	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	13,450	100.00	2.00	100.00	3.3	100.00	1.95	100.00
1. Lead concentrate	154	1.15	38.2	21.87	8.2	2.84	18.7	10.98
2. Lead concentrate	621	4.62	9.8	22.62	7.2	10.07	20.8	49.25
3. Lead concentrate	138	1.03	55.0	28.22	11.2	3.48	6.7	3.52
4. Zinc concentrate	520	3.87	4.8	9.28	52.0	60.90	4.3	8.52
5. Zinc middling	205	1.52	8.2	6.25	19.7	9.09	7.4	5.78
6. Iron underflow	240	1.78	6.00	5.35	8.4	4.54	5.6	5.12
7. Sand tailing	3,785	28.14	.15	2.11	.4	3.41	.3	4.33
8. Tailing	7,787	57.89	.10	2.89	.35	6.14	.4	11.87
Footing	13,450	100.00	.....	98.59	.....	100.47	.....	99.37

Product.	Weight of products (grams)	Per cent of total weight.	Copper.		Gold.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	13,450	100.00	0.30	100.00	0.02	100.00	1.21	100.00
1. Lead concentrate	154	1.15	.80	3.05	.09	5.16	11.33	10.72
2. Lead concentrate	621	4.62	2.42	37.25	.36	83.27	8.28	31.58
3. Lead concentrate	138	1.03	.52	1.77	.12	6.17	19.06	16.16
4. Zinc concentrate	520	3.87	2.78	35.82	.02	3.88	5.66	18.08
5. Zinc middling	205	1.52	1.88	9.55	.02	1.53	6.56	8.26
6. Iron underflow	240	1.78	.70	4.16	Trace.	.....	3.98	5.87
7. Sand tailing	3,785	28.14	.04	3.75	None.	.....	.12	2.79
8. Tailing	7,787	57.89	.03	5.78	Trace.	.....	.14	6.68
Footing	13,450	100.00	.....	100.13	.....	100.01	.....	100.16

TABLE 17.—*Résumé of locked test of Columbus ore.*

Product.	Weight of products (grams)	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	13,450	100.00	2.0	100.00	3.3	100.00	1.95	100.00
Lead concentrate	913	6.80	21.42	72.71	8.0	16.39	18.3	63.75
Zinc concentrate	520	3.87	4.8	9.28	52.0	60.90	4.3	8.52
Zinc middling	205	1.52	8.2	6.25	19.7	9.09	7.4	5.78
Tailing	11,812	87.81	.24	10.35	.53	14.09	.47	21.32

Product.	Weight of products (grams)	Per cent of total weight.	Copper.		Gold.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	13,450	100.00	0.30	100.00	0.02	100.00	1.21	100.00
Lead concentrate	913	6.80	1.86	41.07	.28	94.60	10.42	58.48
Zinc concentrate	520	3.87	2.78	35.82	.02	3.87	5.66	18.08
Zinc middling	205	1.52	1.88	9.55	.02	1.53	6.56	8.26
Tailing	11,812	87.81	.05	13.69	Trace.	.....	.21	15.34

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SUNNYSIDE, AT EUREKA.

The Sunnyside mine and mill at Eureka are operated by the Sunnyside Mining & Milling Co., the mine being now (1921) the most important in the Silverton district, and the company has probably done far more experimenting on so-called differential flotation of these sulphides than any other company in Colorado. It has evolved a flotation separation for its ore that is fairly successful but is not wholly satisfactory.

Table 18 shows the details of a locked test of the Sunnyside ore, the procedure being much the same as that applied to the Columbus ore (Fig. 4), although the making, in this test, of a sand tailing was not feasible. Another exception will be mentioned later.

By tabling the part of the feed coarser than 120-mesh, 18.14 per cent of all the lead was recovered. Coal tar, cresylic acid, and sodium sulphide were used to make lead concentrate 2. This product is low in zinc content (6 per cent) and contains 28.03 per cent of the copper, 10.92 per cent of the gold, and 17.95 per cent of silver.

Lead concentrates 3 and a are instructive because of their high precious metal content. Figure 4 shows the procedure for getting concentrate 3. The procedure for making a is the other exception mentioned above. The roughing run in the dichromate solution was stopped a short time before the zinc overflow was finished, and the charge remaining in the machine was tabled.

The resulting concentrate was high in lead and precious metals. It is fortunate that the flotation of the gold and silver was so retarded in the dichromate solution, for this suggested advantageous changes in the flow sheet, which are discussed later (p. 54).

As 6, iron underflow, runs about $16 per ton in lead and precious metals, it should have been tabled; it contains 7.33 per cent of the gold and 9.87 per cent of the silver. These would add appreciably to the recoveries tabulated. The tailing is higher in silver (0.85 ounce per ton) than expected; before sodium sulphide was used tailings of less than half this silver content could be made easily. However, the added loss is not prohibitive.

Table 19 summarizes the results. The lead recovery was 79.45 per cent. The zinc middling would bring the zinc recovery up to 80 per cent and maintain a grade of 54 per cent zinc.

The consumption of sodium sulphide and dichromate was 3 pounds and 2 pounds, respectively, per ton of mill feed. The grinding was to 120 mesh except for some of the lead which was recovered in a coarse condition.

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TABLE 18.—*Details of locked test of Sunnyside ore.*

Product.	Weight of products (grams)	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	10,000	100.00	5.6	100.00	7.6	100.00	2.45	100.00
1. Lead concentrate	166	1.66	61.2	18.14	5.5	1.20	8.4	5.69
2. Lead concentrate	566	5.66	28.9	29.21	6.0	4.47	19.1	44.12
3. Lead concentrate	220	2.20	63.0	24.76	9.1	2.66	5.4	4.87
a. Lead concentrate	75	.75	54.8	7.34	5.1	.52	10.4	3.20
6. Iron underflow	380	3.80	3.4	2.31	7.0	3.50	7.5	11.65
4. Zinc concentrate	1,029	10.29	5.0	9.19	54.0	73.13	2.8	11.76
5. Zinc middling	260	2.60	8.4	3.90	30.7	10.52	5.7	6.05
8. Tailing	7,300	73.00	.4	5.21	.4	3.85	.4	11.98
Footing	9,996	99.96	.....	100.06	.....	99.85	.....	99.32

Product.	Weight of products (grams)	Per cent of total weight.	Copper.		Gold.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	10,000	100.00	0.42	100.00	0.057	100.00	5.11	100.00
1. Lead concentrate	166	1.66	.42	1.66	.25	7.28	18.31	5.95
2. Lead concentrate	566	5.66	2.08	28.03	.11	10.92	16.21	17.95
3. Lead concentrate	220	2.20	.44	2.30	.56	21.61	33.60	14.46
a. Lead concentrate	75	.75	.42	.75	1.38	18.16	34.40	5.05
6. Iron underflow	380	3.80	.26	2.40	.11	7.33	13.27	9.87
4. Zinc concentrate	1,029	10.29	2.06	50.47	.08	14.44	11.72	23.60
5. Zinc middling	260	2.60	.98	6.06	.16	7.30	21.44	10.91
8. Tailing	7,300	73.00	.05	8.69	.01	12.81	.85	12.14
Footing	9,996	99.96	.....	100.36	.....	99.85	.....	99.93

TABLE 19.—*Résumé of locked test of Sunnyside ore.*

Product.	Weight of products (grams)	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	10,000	100.00	5.6	100.00	7.6	100.00	2.45	100.00
Lead concentrate	1,027	10.27	43.3	79.45	6.5	8.80	12.8	57.88
Zinc concentrate	1,029	10.29	5.0	9.19	54.0	73.13	2.8	11.76
Zinc middling	260	2.60	8.4	3.90	30.7	10.52	5.7	6.05
Iron underflow	380	3.80	3.4	2.31	7.0	3.50	7.5	11.65
Flotation tailing	7,300	73.00	.4	5.21	.4	3.85	.4	11.98

Product.	Weight of products (grams)	Per cent of total weight.	Copper.		Gold.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	10,000	100.00	0.42	100.00	0.057	100.00	5.11	100.00
Lead concentrate	1,027	10.27	1.34	32.74	.32	58.00	21.60	43.41
Zinc concentrate	1,029	10.29	2.06	50.47	.08	14.44	11.72	23.60
Zinc middling	260	2.60	.98	6.06	.16	7.30	21.44	10.91
Iron underflow	380	3.80	.26	2.40	.11	7.33	13.27	9.87
Flotation tailing	7,300	73.00	.05	8.69	.01	12.81	.85	12.14

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It would have been a gross oversight not to investigate the very hard and nearly barren gangue (rhodonite), much of which can be removed before it is ground to flotation size. Accordingly a sample was stage crushed through 40-mesh and the part coarser than 120-mesh tabled to make a sand tailing one-third of the original sample being thus discarded. This discard assayed 0.4 per cent lead, 1.7 per cent zinc, 0.12 per cent copper, trace gold, and 1.06 ounces silver. The zinc and silver content were too high because, during the tabling, sulphides were carried by film suspension across the table and into the gangue. A sample caught for analysis assayed 4.4 per cent lead, 34.6 per cent zinc, 11.1 per cent iron, 2.80 per cent copper, 0.20 ounce gold, and 6.62 ounces silver. Although this loss could not be prevented on the miniature table, it could be eliminated (for grains coarser than 120 mesh) on one of commercial size. Substitution of tabling for the grinding of this very hard gangue, would be a considerable improvement.

BLACK BEAR, AT TELLURIDE.

The Black Bear mine and mill, near Telluride, are operated by the Colorado-Superior Mining Co.

Table 20 shows that the sample of ore tested is of such grade, the cost of grinding must be reduced to a minimum. Accordingly the first test was to make a sand tailing on the miniature Wilfley table, thereby discarding a part of the mill feed and grading up the remainder.

By stage crushing through 30-mesh and tabling the part remaining on 120-mesh, 36 per cent of the original ore was eliminated with an allowable loss. To consider alone the bare analyses of this sand tailing is not sufficient, as it should be borne in mind that it is the most refractory part of the mill feed both as to grinding and as to extraction and grade of products derived. The outstanding characteristic of this ore is that the content of copper (chalcopyrite) content is very high as compared to that of zinc and if not removed will reduce the grade of the zinc concentrate so much that it will not be marketable.

Table 21 covers the series of runs made on minus 80-mesh material to determine the grade of products that could be made by using stated amounts of sodium sulphide with the mixture of coal tar and cresylic acid, the object being to float the copper, gold, silver, and fine lead in advance of the zinc. The results were decidedly encouraging but the sample was exhausted before the requisite amount of sodium sulphide was determined. The floatability of the lead, copper, and precious metals with sodium sulphide is remarkable when the mesh is considered. It is likely that 3 grams sodium sulphide (6 pounds per ton) would give a very desirable product. Flotation could then be continued to make a zinc product (and tailing) to be tabled for the

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recovery of more lead, gold, and silver; this would give a very high recovery of lead. Tailings made in a manner similar to that stated above contained only about 5 per cent of the economic minerals. The zinc product could be cleaned and "graded-up" by the procedure used for the other ores.

In a later test on the Black Bear ore the tar mixture was replaced by xylidin. The results were very promising; the overflow was cleaner and the chalcopyrite floated more freely.

TABLE 20.—*Tabling of coarse product in Black Bear ore for sand tailing.*

Product.	Weight of products (grams)	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Ore	2,000	100.00	2.3	100.00	4.2	100.00	4.6	100.00
Sand tailing	720	36.00	.15	2.35	.4	3.31	.9	7.04

Product.	Weight of products (grams)	Per cent of total weight.	Copper.		Gold.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Ore	2,000	100.00	1.04	100.00	0.11	100.00	2.12	100.00
Sand tailing	720	36.00	.09	3.12	.04	13.10	.28	4.75

TABLE 21.—*Comparative flotation tests of Black Bear ore (—80-mesh) to determine desirable amounts of sodium sulphide.*

Product.	Weight of products (grams)	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed (No Na_2S)	1,000	100.0	2.6	100.00	4.3	100.00	4.7	100.00
First grab	115	11.5	17.4	77.00	14.7	39.32	12.0	29.36
Second grab	40	4.0	3.8	5.84	16.5	15.35	8.4	7.81
Total	155	15.5	.....	82.84	.....	54.67	.....	37.17
First grab (1 gram Na_2S)	120	12.0	12.1	55.84	8.0	22.32	17.0	43.41
Second grab	65	6.5	5.3	13.25	8.8	13.31	14.6	20.19
Total	185	18.5	.....	69.09	.....	35.63	.....	63.60
First grab (2 grams Na_2S)	127	12.7	13.4	65.45	5.0	14.77	17.6	47.55
Second grab	65	6.5	7.7	19.25	16.6	25.09	13.0	18.00
Total	192	19.2	.....	84.70	.....	39.86	.....	65.55

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TABLE 21.—*Comparative flotation tests of Black Bear ore (—80-mesh) to determine desirable amounts of sodium sulphide*—Continued.

Product.	Weight of products (grams)	Per cent of total weight.	Copper.		Gold.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed (No Na_2S)	1,000	100.0	1.04	100.00	0.11	100.00	2.00	100.00
First grab	115	11.5	7.40	81.83	.42	43.90	14.50	83.40
Second grab	40	4.0	1.92	7.39	.27	9.90	5.55	11.10
Total	155	15.5	.....	89.22	.....	53.80	.....	94.50
First grab (1 gram Na_2S)	120	12.0	4.84	55.61	.....	.....	.....	.....
Second grab	65	6.5	3.12	19.50	.....	.....	.....	.....
Total	185	18.5	.....	75.11	.....	.....	.....	.....
First grab (2 grams Na_2S)	127	12.7	5.16	63.01	.30	34.70	9.29	58.80
Second grab	65	6.5	4.86	30.38	.48	28.40	9.54	31.00
Total	192	19.2	.....	93.39	.....	63.10	.....	89.80

TOMBOY, AT TELLURIDE.

The Tomboy mine and mill, near Telluride, are operated by the Tomboy Gold Mines Co. (Ltd.). The ore is like the Black Bear ore just described. In making the products shown in Table 22 the same procedure was followed as in testing the Columbus ore in Figure 4.

One-fourth of the mill feed was discarded by making a sand tailing (7, Table 22), and by tabling the mixed sulphide thus obtained, 31.97 per cent of the lead was recovered.

The large amount of lead and precious metals recovered in (1) lead concentrate and the low metal content of the sand tailing indicated that the grinding could have been coarser—that is, to 40-mesh instead of 50-mesh—thus increasing the amount of coarse lead and decreasing the expense of grinding.

The grades of the lead concentrates 2a and 2b indicate the required amount of sodium sulphide for the best results; the zinc in 2b assays only 9 per cent, but assays 16 per cent in 2a. This improvement resulted from increasing the amount of sodium sulphide from 3 grams to 4 grams in the 4-liter machine. As there were nine runs and 6.5 gram-tons of mill feed, the consumption of sodium sulphide when using 4 grams would amount to 5.5 pounds per ton of mill feed. Only one drop of the coal tar and cresylic acid mixture was required for each run. Table 23 shows the lead recovery was 84.79 per cent.

There was only enough of the zinc product for two runs in locked-test fashion. Each run consisted of one roughing and three cleanings. The dicromate required in the successive steps of each run (4-liter machine) was 1.5 grams, 0.5 gram, 0.5 gram, and 0.25 gram; the solu-

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tion used in the roughing was retained in the circuit; the dichromate required amounted to 0.84 pound per ton of mill feed.

The zinc concentrate was not satisfactorily graded up because of the low zinc content of the zinc middling; a heavier middling should have been made. As much better results can be obtained, it is unfortunate that time for a further study of the ore was not available.

The marked segregation of precious metals in 3, lead concentrate, is in accord with tests of other San Juan ores, the precious metals being more easily tabled into a lead product from this fine material than from the material coarser than 120-mesh.

In a later test of the Tomboy ore the tar mixture was replaced by xylidin, and in another test by a very small amount of steam-distilled pine oil. Results were encouraging, and the investigation should be carried further.

TABLE 22.—*Details of locked test of Tomboy ore.*

Product.	Weight of products (grams)	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	13,000	100.00	3.0	100.00	3.4	100.00	4.5	100.00
1. Lead concentrate	271	2.08	46.0	31.97	2.8	1.72	18.9	8.77
2a. Lead concentrate	425	3.27	15.2	16.57	16.0	15.38	13.5	9.82
2b. Lead concentrate	586	4.51	14.0	21.04	9.0	11.93	16.8	16.81
3. Lead concentrate	103	.79	57.6	15.21	1.8	.42	14.6	2.51
4. Zinc concentrate	570	4.38	2.2	3.22	41.6	53.62	11.1	10.83
5. Zinc middling	240	1.85	3.2	1.97	7.4	4.02	19.5	8.01
6. Iron underflow	260	2.00	7.1	4.74	6.0	3.52	18.9	8.41
7. Sand tailing	3,365	25.89	.15	1.29	.4	3.05	1.5	8.64
8. Tailing	6,950	53.46	.25	4.46	.4	6.31	2.2	26.14
Footing	12,770	98.23	.....	100.47	.....	99.87	.....	100.05

Product.	Weight of products (grams)	Per cent of total weight.	Copper.		Gold.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	13,000	100.00	0.54	100.00	0.195	100.00	3.25	100.00
1. Lead concentrate	271	2.08	.40	1.54	2.67	28.50	21.41	13.75
2a. Lead concentrate	425	3.27	3.94	23.85	.61	10.21	17.49	17.62
2b. Lead concentrate	586	4.51	3.46	28.88	.56	13.72	15.18	21.08
3. Lead concentrate	103	.79	.20	.29	5.23	21.22	34.21	8.35
4. Zinc concentrate	570	4.38	3.28	26.63	.32	7.18	9.08	12.27
5. Zinc middling	240	1.85	1.72	5.88	.24	2.27	8.34	4.75
6. Iron underflow	260	2.00	1.56	5.78	.33	3.38	12.21	7.55
7. Sand tailing	3,365	25.89	.06	2.87	.04	5.31	.60	4.78
8. Tailing	6,950	53.46	.04	3.96	.03	8.21	.60	9.88
Footing	12,770	98.23	.....	99.68	.....	100.00	.....	100.03

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TABLE 23.—*Résumé of locked test of Tomboy ore.*

Product.	Weight of products (grams)	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	13,000	100.00	3.0	100.00	3.4	100.00	4.5	100.00
Lead concentrate	1,385	10.66	23.9	84.79	9.4	29.45	16.0	37.98
Zinc concentrate	570	4.38	2.2	3.22	41.6	53.62	11.1	10.83
Zinc middling	240	1.85	3.2	1.97	7.4	4.02	19.5	8.01
Iron underflow	260	2.00	7.1	4.74	6.0	3.52	18.9	8.41
Tailing	10,315	79.35	.22	5.75	.4	9.36	2.0	34.77

Product.	Weight of products (grams)	Per cent of total weight.	Copper.		Gold.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.	Asay (ounces per ton).	Per cent of feed content.
Feed	13,000	100.00	0 54	100.00	0.195	100.00	3.25	100.00
Lead concentrate	1,385	10.66	2.77	54.56	1.34	73.65	18.52	60.80
Zinc concentrate	570	4.38	3.28	26.63	.32	7.18	9.08	12.27
Zinc middling	240	1.85	1.72	5.88	.24	2.27	8.34	4.75
Iron underflow	260	2.00	1.56	5.78	.33	3.38	12.21	7.55
Tailing	10,315	79.35	.05	6.83	.033	13.52	.60	14.66

MARION, AT RYE.

Two samples of Marion ore were examined early in 1920; they were used in preliminary tests that revealed no plan of treatment that seemed capable of satisfactory development.

The first sample assayed 0.3 per cent lead, 27.3 per cent zinc, 7.8 per cent iron, and 2.86 per cent copper (chalcopyrite). The second assayed 2.4 per cent lead, 7.6 per cent zinc, 4.8 per cent iron, and 2.12 per cent copper.

In the light of later work, it seems that the method applied to the San Juan ores would be as successful for the Marion ore.

Mica is one of the troublesome constituents of this ore.

PECOS, AT VALLEY RANCH, N. MEX.

The Pecos ore, like the Marion, contains mica, which is the most troublesome constituent, and the content of base metal is similar. It was taken as a substitute for the Marion because another sample of the Marion could not be easily procured.

The time allowed for this work was so short that in order to complete a locked test only a very brief preliminary investigation could be made. The procedure followed was nearly the same as that applied to the Columbus ore (Fig. 4). The Pecos ore is so high grade that the making of a sand tailing on a table would not be economically feasible except for the mica causing trouble in flotation. Tabling removed 7.33 per cent of the mill feed as mica tailing. As

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the mica causes trouble in flotation and is difficult to grind, this tabling is desirable, since some of the mica can be tabled out and discarded cheaper than it can be ground. Furthermore, there is a remote possibility of finding a market for the mica.

The results shown in Table 24 require no discussion except (a) iron concentrate. Tests had shown that the flotation of the galena and precious metals was retarded in the dichromate solution. In order to take advantage of the segregation brought about by this retardation, the roughing flotation in dichromate was stopped before the run was finished and the underflow was tabled. A concentrate called (a) iron concentrate was obtained. The high content of lead and precious metal in this concentrate shows that a similar procedure in a mill would add to the recoveries. The exact imitation in the mill of what was done in the laboratory would not be feasible, but a scheme could be substituted that would be simpler and more effective. This will be discussed later. The tailing from the tabling was put back in the flotation machine and the run finished.

TABLE 24.—*Detail of locked test of Pecos ore.*

Product.	Weight of products (grams).	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	10,000	100.00	4.08	100.00	18.9	100.00	11.3	100.00
1. Lead concentrate	296	2.96	23.0	16.68	5.4	.84	29.9	7.83
2. Lead concentrate	1,542	15.42	12.5	47.16	10.8	8.79	16.8	22.95
3. Lead concentrate	109	1.09	38.0	10.15	3.5	.20	23.7	2.29
Iron concentrate from tailings	105	1.05	3.3	.84	3.2	.18	40.5	3.76
(a) Iron concentrate	92	.92	10.2	2.25	2.3	.11	38.4	3.13
4. Zinc concentrate	2,904	29.04	2.1	14.95	47.2	72.34	6.9	17.73
5. Zinc middling	426	4.26	2.4	2.50	22.4	5.09	15.1	5.69
6. Iron underflow	1,380	13.80	.7	2.36	12.0	8.75	20.2	24.67
7. Mica tails	733	7.33	.4	.71	3.5	1.35	3.4	2.21
8. Tailing	2,390	23.90	.4	2.34	1.8	2.27	4.6	9.73
Footing	9,977	99.77	.....	99.94	.....	99.92	.....	99.99

Product.	Weight of products (grams).	Per cent of total weight.	Copper.		Gold.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	10,000	100.00	1.10	100.00	0.141	100.00	4.13	100.00
1. Lead concentrate	296	2.96	.40	1.07	.73	15.41	14.21	10.18
2. Lead concentrate	1,542	15.42	2.84	39.91	.14	15.26	8.33	31.13
3. Lead concentrate	109	1.09	.22	.22	2.89	22.50	25.83	6.82
Iron concentrate from tailings	105	1.05	.16	.15	.56	4.20	5.08	1.29
(a) Iron concentrate	92	.92	.28	.23	.44	2.86	7.02	1.56
4. Zinc concentrate	2,904	29.04	1.76	46.46	.08	16.55	4.28	30.09
5. Zinc middling	426	4.26	.90	3.48	.08	2.39	3.36	3.46
6. Iron underflow	1,380	13.80	.38	4.77	.10	9.90	2.50	8.35
7. Mica tails	733	7.33	.21	1.39	.09	4.98	1.13	2.00
8. Tailing	2,390	23.90	.12	2.60	.04	6.81	.89	5.15
Footing	9,977	99.77	.....	100.28	.....	100.86	.....	100.03

A résumé of the results appears in Table 25.

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TABLE 25.—*Résumé of locked test of Pecos ore.*

Product.	Weight of products (grams).	Per cent of total weight.	Lead.		Zinc.		Iron.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Feed	10,000	100.00	4.08	100.00	18.9	100.00	11.3	100.00
Lead concentrate	2,144	21.44	14.7	77.08	8.9	10.12	21.1	39.96
Zinc concentrate	2,904	29.04	2.1	14.95	47.2	72.34	6.9	17.73
Zinc middling	426	4.26	2.4	2.50	22.4	5.09	15.1	5.69
Iron underflow	1,380	13.80	.7	2.36	12.0	8.75	20.2	24.67
Tailing	3,123	31.23	.4	3.05	2.2	3.62	4.3	11.94

Product.	Weight of products (grams).	Per cent of total weight.	Copper.		Gold.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	10,000	100.00	1.10	100.00	0.14	100.00	4.13	100.00
Lead concentrate	2,144	21.44	2.13	41.58	.39	60.23	9.73	50.98
Zinc concentrate	2,904	29.04	1.76	46.46	.08	16.55	4.28	30.09
Zinc middling	426	4.26	.90	3.48	.08	2.39	3.36	3.46
Iron underflow	1,380	13.80	.38	4.77	.10	9.90	2.50	8.35
Tailing	3,123	31.23	.14	3.99	.05	11.79	.94	7.15

The consumption of sodium sulphide and dichromate was 4 pounds and 4.5 pounds, respectively, per ton of mill feed; the amount of Cleveland Cliffs No. 2 oil was 1 pound per ton. The sulphuric acid required was always about one-fifth the weight of the dichromate.

COMMENTS ON TESTS OF COMPLEX SULPHIDE ORES.

PERSONNEL AND METHODS.

The foregoing pages give the result of an examination of 18 ores of the type commonly known as complex sulphides. C. O. Anderson was the only member of the staff who conducted tests during the whole period of 10 months that was allotted to the work. J. C. Williams and Max Shapiro assisted during the first six months and five months, respectively. Mr. Williams devoted the remaining part of the period to the oxidized ores. Russell R. Bryan assisted for one week and brought valuable information on the practice at the Sunnyside mill. A. D. Storke gave valuable assistance during the last six weeks of the work.

Two months were spent in research, leaving eight months for the examination of the ores. In consequence there was not time for making exhaustive tests of any ore or group of ores. The report should be considered as a unit rather than as a series of completed tests.

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The methods applied were combinations of gravity concentration and flotation. The work was done in full appreciation of the fact that pitfalls are sometimes met between the laboratory and the mill; to apply gravity concentration wherever possible is the best safeguard, and it minimizes the inconsistencies of so-called differential flotation mentioned on page 7. Gravity concentration was used to recover not only galena, but also a considerable part of the gold and silver.

FLOTATION REAGENTS.

The status of flotation in Colorado is not creditable; one operator has gone so far as to say that flotation has been a dismal failure in this State. On the other hand, this work has proved surprising by showing that the laboratory work can be systematized and the same process applied, with slight variations, to all the ores. If flotation had required the guessing considered necessary in its earliest stages of development, this work could not have been done. The authors do not claim that the oils used were the best, but what had to be used in order to accomplish the tasks assigned within the period allotted. Although the selection of oils requires more study, the same is true to a less degree of the acid dichromate and sodium sulphide. The chemistry of the dichromate solution is fairly well understood and the study of sodium sulphide has been begun. Sodium sulphide is a reagent commonly used for making oxides of lead and copper floatable, and its failure to make oxidized zinc floatable is well known. These facts coordinate nicely with its effect on sulphides, for it makes the galena and chalcopyrite float in advance of sphalerite. If these sulphides are regarded as coated with incipient oxide films the parallel is complete.

SODIUM SULPHIDE.

The chemical consumption of sodium sulphide in the presence of galena, sphalerite and pyrite is appreciable, and can be accounted for only by an oxide-film theory. Further study of the chemistry of these reactions must be made, and the rate of oxidation in aerated solutions and the deportment of the products of oxidation must be determined. For example, it is likely that the sulphide should be added immediately before flotation begins, and that adding it in advance, for instance in the ball mill, would be detrimental. There is some evidence that the adding of the sodium sulphide and tar to the dilute pulp just before flotation is not only permissible but is desirable. This would be a great advantage as compared with the common practice of beating tar into a thick pulp. Of course, it is too much to expect that sodium sulphide will give good results in a mill feed

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that is contaminated with soluble natural salts; the ores to which it was applied in these tests were compact and comparatively unaltered.

Sodium sulphide increases the floatability of pyrite; consequently its addition may be undesirable when the pyrite is barren. If the content of the pyrite in the ore is high as compared to that of lead, the assay per cent of lead in the lead froth will be very low.

TARS.

Tars as flotation reagents are commonly regarded as making the froths too siliceous. This objection holds for the work considered here. In this work it was desired to have the tar make a froth carrying not only the finest grains of galena, but also a goodly percentage of the chalcopyrite and precious metals. The tar fell short of the requirements as to chalcopyrite and silver minerals, and a substitute should be sought. Steam-distilled pine oil and xylidin were tried with the sodium sulphide and the results were very encouraging. As is well known, tar is a poor frothing reagent. Its beneficial effects in this work would have been overlooked if the first tests had been made in the 9-liter instead of the 4-liter hollow-shaft machine. In these two machines the quantity of air supplied is the same; it proved entirely inadequate for the 9-liter charge but ample for the 4-liter charge.

ACID DICHROMATE.

Acid dichromate deserves serious consideration as a flotation reagent. In 1919 the average market price of chemically pure sodium dichromate was 12 cents per pound; early in 1920 the price advanced considerably, but in November of that year it dropped to 8 cents. A commercial grade carrying some available acid is not only allowable but desirable, and it should be obtainable at a lower price. To approximate the amount of dichromate required for a given ore the percentage of zinc may be divided by four and the quotient taken as pounds of dichromate per ton of mill feed. As the dichromate is easily soluble in water the acid solution can be made up in tanks for each shift..

In the beginning of the work the dichromate was used to prevent the galena from floating with the sphalerite, but its positive effect in preventing pyrite from floating with the sphalerite was soon realized and the effect on galena was lost sight of. Only at the end of the work did the remarkable properties of this reagent begin to be appreciated. In addition to the positive "deadening" effect upon pyrite, the flotation of the galena, gold, and some of the silver minerals is retarded enough so that the recoveries of these minerals may be increased by a better manipulation than is indicated by the foregoing notes. After

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a review of the manipulation above referred to, the proposed improved procedure will be discussed.

The practice of cleaning the zinc product in the dichromate solution was to continue the roughing run (see definition of terms, p. 16) until all the sphalerite had come over in the froth and a pyrite product was drawn to make an iron underflow. The zinc froth was returned to the machine and flotation was repeated as often as desirable, but instead of an iron underflow practically free from zinc being made, zinc-iron middlings were made by stopping the flotation before all the zinc had come over in the froth; the overflow from the last cleaning constituted the zinc concentrate. The zinc middlings were put with the new charge to constitute the subsequent run. One of the weak points in this method was that the zinc middlings reentered flotation without any intermediate treatment, although a middling should have some kind of treatment to change its nature before it reenters the circuit from which it came. Ordinarily, grinding is practiced, but here gravity concentration, not grinding, is required to remove galena, gold, and silver minerals.

The procedure should be modified as follows: The first part of the zinc froth formed during the roughing of the zinc product should be taken to make a zinc concentrate corresponding to the concentrate made on the first cell of a multicell commercial machine. The rest of the overflow should be passed over concentrating tables to remove galena, gold, and silver minerals. Some pyrite would also be removed. The table tailing, enriched in zinc, would be sent back (after grinding) to the head of the rougher. An examination of the analyses of the zinc middlings and iron underflows in the foregoing tabulations will show the importance of these changes. To table the zinc middling after each flotation run would be a very tedious method of locked testing, but it is the only means of imitating mill procedure.

The dichromate solution seems to have some corroding effect on the linoleum of the concentrating table. This would require watching in commercial work.

A very small amount of oil is required for cleaning the zinc product because the sphalerite has previously been oiled amd floated. No. 350 Pensacola Tar & Turpentine Co. oil and No. 2 Cleveland Cliffs oil are entirely satisfactory. Others have not been tried.

Slight attention to soluble salts and gangue has been given in this report. They seemed to cause very little trouble and, therefore, were neglected.

STEPS IN FLOTATION.

The literature on flotation carries the idea that each ore requires a separate and special process. Thus it would seem that we have no "flotation process," only certain "steps" which are patented and

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for which royalties are exacted. Hence it was a surprise to find that the laboratory work on the Colorado ores could be reduced to a "process." The following generalizations show to what degree this was effected:

1. Crushing to 120-mesh for the roughing flotation of the mill feed is ample.

2. By a roughing flotation the extraction of the economic minerals from the mill feed is seldom below 95 per cent.

3. Low-grade tailings may be obtained by using either one of the following two oils: No. 350 Pensacola Tar & Turpentine Co. oil (with sodium bicarbonate) or No. 2 Cleveland Cliffs oil.

4. When it is desired to float fine galena, chalcopyrite, and precious metals in advance of the sphalerite, sodium sulphide with one of the following reagents is effective: A mixture of coal tar and cresylic acid, steam-distilled pine oil, or xylidin. (The Akron ore is an exception, probably on account of soluble salts.) The sphalerite will float freely when No. 2 Cleveland Cliffs oil is used.

5. Acid dichromate when applied to a product that has been bulk-floated (with the proper reagents) has a positive "deadening" effect on pyrite and an advantageous deadening effect on galena and precious metals. These minerals may be made concentrated effectively by gravity methods where their segregation is advantageous.

6. Unfortunately, chalcopyrite floats in the acid dichromate even more freely than does sphalerite. This points to a method for grading up the copper in chalcopyrite-pyrite ores.

7. The same flow sheet, with slight exceptions, had been followed in all the laboratory work.

After the laboratory, comes the pilot mill and then the commercial mill. The last offers the only proof of the effectiveness of the others.

The results of this rapid laboratory survey of the metallurgical deportment of the Colorado complex sulphide ores lends hope that a satisfactory process of treatment will be forthcoming where an ore body is developed enough to justify building an adequate mill combining gravity concentration and flotation.

TESTS OF OXIDIZED ORES.

For convenience the samples of oxidized ores were divided into two groups: J. P. Bonardi made an examination of the Esperanza, Monon, Manitoba, Commodore, and Bachelor ores; J. C. Williams made an examination of the Pony Express, Senorita, Black Girl, Wedge, Robert E. Lee, and Dunkin ores.

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ESPERANZA, AT DEL NORTE.

The Esperanza mine, near Del Norte, is operated by George D. Nickel.

According to figures submitted by operators a carload of the Esperanza selected ore shipped to a smelter gave the following analysis:

Analysis of Esperanza ore.

Iron	4.5 per cent.
MnO_2	14.4 per cent.
Zinc	.5 per cent.
Sulphur	.9 per cent.
SiO_2	61.6 per cent.
Gold	.08 ounce per ton.
Silver	33.77 ounces per ton.

The sample used in the experimental work gave the following analysis:

Iron	3.30 per cent.
Free MnO_2	5.45 per cent.
Total manganese	5.40 per cent.
Sulphur	.30 per cent.
Gold	Trace.
Silver	21 ounces per ton.

These analyses and the fact that thorough cyanidation gave less than 50 per cent of the silver, and the fact that dissolving the MnO_2 in SO_2 water and washing followed by cyanidation gave a very high extraction of the silver (98.5 per cent) show that the ore is, without doubt, a refractory manganese-silver ore.

The Caron-Clevenger process is adapted particularly to such ores. The early work on this process was done at the Golden station of the Federal Bureau of Mines, and, as the equipment was still available, it was used in this investigation. In the Caron-Clevenger process, the crushed ore is subjected to a reducing gas (producer gas, water gas, wood gas, or any other reducing agent) as is described in United States Patent No. 1298454, assigned to the Research Corporation of New York. For the Esperanza ore carbon monoxide, made by the incomplete combustion of charcoal, was passed through the revolving ore-chamber, after the ore had been heated to the proper temperature (600° to 800° C.), to reduce the manganese dioxide. During the run the temperature was maintained at about 800° C. When the excess gas at the outlet of the drum or furnace could be ignited by a match, the charge was fully reduced; then the gas was passed through for a short period, after which the charge was allowed to cool in a reducing atmosphere.

A small amount (1,000 grams) of minus 4-mesh material was reduced in the experimental Caron furnace by carbon monoxide gas

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at 800° C. Passage of the gas over the hot ore for 45 minutes was required before the material was fully reduced, after which the gas was passed through the charge for 15 minutes and then the drum was lifted out of the furnace and allowed to cool in a reducing atmosphere under the gas pressure from the gasometer. The material lost 75 grams. The reduced ore was crushed through 100-mesh, and 300 grams of it (no free MnO_2 present) was thoroughly ground with cyanide solution and lime. Then the ground pulp was agitated for 17 hours in a separatory funnel by air drawn through the charge by an aspirator pump. The silver extraction after this treatment was 90.9 per cent with a cyanide consumption of 7.7 pounds NaCN per ton.

Runs were then made to decrease, if possible, the excessive cyanide consumption. In one test a part of the minus 100-mesh reduced ore was washed with water, then with lime water and filtered. The residue was cyanided, the silver extraction being 85.6 per cent, and the cyanide consumption 5.3 pounds per ton. This test was repeated except that the ore was thoroughly ground before cyaniding; the silver extraction was 90.0 per cent and the cyanide consumption was 3.7 pounds per ton. Evidently cyanidation of the reduced ore, without washing, for example, to remove cyanicides, does not result in low cyanide consumption; however, a water wash and thorough grinding decrease the subsequent cyanide consumption and maintain the extraction.

The results show that the Esperanza ore is amenable to cyaniding in fresh solutions, provided it is first reduced either by the Caron-Clevenger process or by SO_2 water. Either method gave a 90 per cent extraction, or better, with a cyanide consumption of about 4 pounds NaCN per ton.

MONON, AT CREEDE.

The Monon mine, at Creede, is owned by the Moon Mining Co. and is leased by the Wabash Mines & Power Co. The sample of Monon ore assayed 13.20 ounces silver per ton. It was crushed through 200-mesh and then given a cyanide test by agitation in a closed Mason jar. The pulp density was three to one, and the strength of the cyanide solution was 3.6 pounds NaCN. Lime was added in excess of its solubility. After 12 hours 89 per cent of the silver was extracted, and the cyanide consumption was 2.6 pounds per ton of ore. After 72 hours, 92.42 per cent of the silver was extracted; the cyanide consumption was 4 pounds, and the lime consumption was 20.8 pounds per ton of ore.

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MANITOBA, AT CREEDE.

The Manitoba mine, near Creede, is operated by the Manitoba Leasing Co. The sample of ore assayed 11.66 ounces silver per ton. It was crushed through 200-mesh and agitated with cyanide solution in a closed jar. The strength of the cyanide solution was 3.6 pounds NaCN, and lime was added in excess of its solubility. The pulp density was three to one. After 12 hours, 72 per cent of the silver was extracted with a cyanide consumption of 4.1 pounds per ton of ore. After 72 hours, 76.4 per cent of the silver was extracted; the sodium cyanide and lime consumption were 5.6 and 21.3 pounds per ton of ore, respectively.

COMMODORE, AT CREEDE.

The Commodore mine, at Creede, is owned by the Commodore Mining Co. The sample of ore assayed 13.24 ounces silver per ton. The procedure in cyaniding was the same as for the Monon and Manitoba ores. After 12 hours, 79 per cent of the silver was extracted with a cyanide consumption of 2.6 pounds NaCN per ton of ore. After 72 hours, 81.40 per cent of the silver was extracted; the consumption of cyanide and of lime was 4 pounds and 20.5 pounds per ton of ore, respectively.

A second sample sent to the laboratory for examination analyzed higher in silver, 18.22 ounces per ton. Cyanidation in the usual manner on minus 150-mesh material gave, after 24 hours, 85.5 per cent extraction of the silver. The sodium cyanide consumption was 3.1 pounds per ton.

BACHELOR, AT CREEDE.

The Bachelor mine, which joins the Commodore mine, is owned by the Bachelor Mining Co. This ore analyzed 7.75 ounces of silver per ton, 3.27 per cent free MnO_2 (7.23 per cent total manganese), and 1.7 per cent iron. Only 30 per cent of the silver could be dissolved by the usual procedure of cyaniding. A preliminary treatment in sulphurous acid made possible better results. The residue from this treatment after being cyanided 24 hours yielded 65 per cent of the silver, with a sodium cyanide consumption of 3.8 pounds per ton of ore.

Because of its free MnO_2, and because of its silver content resisting cyanidation, this ore may be placed in the refractory manganese-silver class.

To determine the behavior of this ore in the Caron-Clevenger process, used for the Esperanza ore (p. 56), 1,000 grams, ground to minus 4-mesh, was reduced in the Caron furnace at a temperature of about 800° C., 45 minutes of continual passage of carbon monoxide gas over the hot ore being necessary for complete reduction. The

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material lost 55 grams. Two hundred grams of the reduced ore (by test, no free MnO_2 present) was ground through 80-mesh, then ground in a laboratory porcelain jar for 4 hours in 600 c. c. of cyanide solution (strength, 3.6 pounds per ton) and 2 grams of lime. The ground pulp was agitated with air in a separatory flask for 65 hours. The silver extraction was 82.2 per cent, and the cyanide consumption was 23 pounds per ton of ore.

Unless remedied, the high consumption of cyanide would render this process prohibitive for this ore, but probably the cyanide consumption can be decreased somewhat. No attempt to reduce it was made because the tonnage of Bachelor ore is small. Reference to the work on the Esperanza ore (p. 56) shows that a high cyanide consumption was obtained after reduction, and that a simple wash with water before cyanidation decreased the consumption about 50 per cent.

PONY EXPRESS, AT CREEDE.

The sample, representative of the ore exposed in the Pony Express mine, Ouray, was furnished by Mr. Clifford E. Wilfley, of Ouray, who said he took the sample personally and believed it truly represented the metallurgical characteristics and the value of the ore exposed. He said that the great importance of the sample was due to its representing a well-known type of ore developed in a number of mines in the north end of the Ouray mining district.

The ore is more or less a replacement of a thick limestone stratum. A sample of the material used gave the following assays:

Au, trace; Ag, 18.12 ounces; Cu, 0.08 per cent; Pb, 0.8 per cent; Fe, 1.2 per cent; CaO, 6.4 per cent; S, 0.26 per cent; Mn, 0.77 per cent.

All the mineral constituents were not determined. At least some of the silver is evidently in the form of a sulphantimonate, as a trace of antimony was identified in the solution from leaching the ore with caustic soda.

Arsenic is also very likely in combination with the silver. The sulphide minerals, present in small amount, include easily recognized galena and pyrite. The iron is largely oxidized, as is the copper, and the ore contains an appreciable amount of oxidized lead as cerussite. The condition in which the manganese exists was not determined.

Screen analysis shows a close association of the silver and lead; each one of the screened products contained almost the same proportional amounts of each metal. Consequently, the silver probably is combined with the lead minerals.

A standard cyanide solution, used in all tests, was made by dissolving about 30 grams of commercial sodium cyanide ("Cyanegg")

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in 16 liters of saturated lime water. This gave a solution strength of 3.6 pounds NaCN and 2.4 pounds CaO per ton of solution. Ore crushed in a disk pulverizer to pass 100-mesh was used as the feed in all tests.

One hundred grams of 100-mesh ore was put in a separatory funnel with 200 c. c. standard cyanide solution and 0.5 gram lime, and agitated by air drawn through the pulp for 16 hours, when 100 c. c. additional cyanide solution was added. The agitation was continued for 48 hours, giving a total time of agitation of 64 hours. After the pulp was filtered, titration of the filtrate indicated a consumption of 4 pounds NaCN and 16 pounds lime per ton of ore. The residue was washed, dried, and assayed, with the following results:

Assay feed, 18.12 ounces Ag; assay tailing, 2.23 ounces Ag; extraction, 87.7 per cent.

In the next test the ore (200 grams) was ground 6 hours in cyanide solution. The pulp was washed into a separatory funnel. The solution was in the ratio of five parts to one of ore, and its cyanide content equivalent to 1.1 pounds NaCN per ton. The pulp was agitated by air for 16 hours and then filtered. (Total time of treatment, 21 hours.) Titration of the filtrate indicated a consumption of 4.3 pounds NaCN per ton of ore. The residue was washed, dried, and assayed, with results as follows:

Assay feed, 18.12 ounces Ag; assay tailing, 1.10 ounces Ag; extraction, 93.9 per cent.

The results of gravity concentration are given in Table 26. Two hundred grams of 100-mesh ore was vanned on a plaque.

TABLE 26.—*Gravity concentration of 100-mesh Pony Express ore.*

Product.	Weight of products (grams).	Per cent of total weight.	Silver.		Lead.	
			Assay (ounces per ton).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.
Concentrate	14	7	134.08	51	6.2	57
Tailing	186	93	9.72	49	.35	43
Total	200	100	.....	100	.....	100

Assay heads, calculated—silver, 18.43; lead, 0.75.
Assay heads, direct—silver, 18.12; lead, 0.80.

The tabulated results show an extraction of 51 per cent of the silver and 57 per cent of the lead, with a ratio of concentration of 14.3:1.

The tailing from vanning was ground through 200 mesh in a porcelain jar with flint pebbles; it was then filtered, washed, dried, and assayed, with this result:

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Recovery:	Ounces, Ag.
Assay feed (vanning tailing)	9.72
Assay cyanide tailing	0.47
	Per cent
Extraction by cyanide (vanner tailing)	95.1
Extraction by cyanide (crude ore)	46.6
Extraction by concentration (crude ore)	51.0
Extraction by combined concentration and cyaniding	97.6

A preliminary wash in a warm caustic soda solution (1 per cent) followed by cyaniding did not show enough advantage to justify its application.

The filtrate from the caustic soda wash contained an appreciable amount of lead and a trace of antimony.

These tests seem to justify further cyanide tests on a larger scale, in order to determine suitable conditions for plant operation. Fine grinding in cyanide solution followed by air agitation might result in a suitable extraction with a consumption of about 3.8 pounds of cyanide and 16 of lime per ton of ore.

Gravity concentration followed by cyaniding of the tailing might also be adapted to this ore if material of the mesh used in these tests is given careful tabling.

In none of the experiments was agitation continued for as many hours as is usually required in plants cyaniding silver ores. Longer agitation might result in a higher extraction.

SENORITA, BLACK GIRL, AND WEDGE, AT OURAY.

The samples of ore from the Senorita, Black Girl, and Wedge mines, Ouray district, were somewhat similar to the Pony Express sample, but their copper content was higher. They assayed as follows:

Assays of ore from Ouray district.

Ore.	Gold (ounces per ton).	Silver (ounces per ton).	Copper (per cent).	Lead (per cent).	Iron (per cent).	Lime (per cent).	Sulphur (per cent).	Manganese (per cent).
Senorita	0.03	26.10	1.06	0.6	1.60	0.30	0.31	0.12
Black Girl	.035	24.48	.96	.5	1.50	1.10	.90	.47
Wedge	.02	32.10	.14	.5	3.70	.12	3.15	Nil.

The Senorita and Black Girl samples were almost completely oxidized; the Wedge contained a considerable amount of sulphide minerals, principally pyrite. The copper in all the samples was largely oxidized and was exceedingly detrimental to cyanidation. Tests showed an excessive consumption of cyanide and a low extraction of the silver. Practically all the cyanide consumption occurred during the first 10 minutes of treatment. The Senorita and Black Girl samples contained small amounts of water-soluble sulphates and no soluble iron; the Wedge showed a large amount of sulphates and ferrous iron.

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Screen analysis showed a very close relationship between the silver and lead. The predominant original economic mineral was argentiferous galena.

It is evident that before cyaniding can be applied the destructive action of the copper must be nullified. Experiments with a saturated solution of sulphur dioxide in water as a solvent for copper were made. Solution was slow and the consumption of sulphurous acid was high. Better results might be obtained if sulphur dioxide gas were passed into the pulp. As it was, not enough copper could be removed to lower to a suitable degree the cyanide consumption.

For a gravity concentration test, a sample of Senorita ore that had been crushed to about 2-mesh was stagecrushed by rolls through 30-mesh, then sized wet on a 100-mesh sieve. One kilogram of the minus 30 plus 100 product was tabled on a laboratory Wilfley table and 200 grams of the minus 100-mesh product was panned.

Table 27 shows the screen analysis of the ore. There was no marked segregation of any of the three metals in either the coarse or the fine material.

TABLE 27.—*Screen analysis of 30-mesh Senorita ore.*

Product.	Weight of products (grams).	Per cent of total weight.	Lead.		Copper.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	2,025	100	0.6	100.0	1.30	100.0	27.88	100.0
−30+100 mesh	1,195	59	.5	49.2	1.22	55.4	21.06	44.6
−100 mesh	830	41	1.0	68.3	1.42	44.7	35.46	52.0
Total	2,025	100	.....	117.5	.....	100.1	.....	96.6

Table 28 gives the results of gravity concentration of the coarse material. The grade of concentrate was fair, as was also the ratio of concentration, but the recovery was very poor. A microscopic examination of the feed gave the impression that the minerals were finely interlocked, but this conclusion was by no means positive, as the minerals could not be identified with certainty.

TABLE 28.—*Gravity concentration of minus 30 plus 100-mesh Senorita ore.*

Product.	Weight of products (grams).	Per cent of total weight.	Lead.		Copper.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	1,000	100.0	0.5	100	1.22	100.0	21.06	100.0
Concentrate	200	20.0	1.0	40	3.02	49.5	57.12	54.2
Tailing	785	78.5	.35	55	.72	46.3	12.44	46.0
Total	985	98.5	.....	95	.....	95.8	.....	100.2

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The grade of the concentrate made from the fine material (see Table 29) was much higher than that made from the coarser material, and the ratio of concentration was doubled. The extraction was much lower, possibly because of the higher ratio of concentration. Evidently the economic minerals were in very small particles, and hence were not liberated in the coarse ore, whereas the free mineral particles in the fine ore were largely washed away from the coarser gangue.

TABLE 29.—*Gravity concentration of minus 100-mesh Senorita ore.*

Product.	Weight of products (grams).	Per cent of total weight.	Lead.		Copper.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	200	100	1.0	100.0	1.42	100.0	35.46	100.0
Concentrate	20	10	3.3	30.4	5.20	36.2	147.02	41.4
Sand tail	130	65	.5	30.8	.78	35.4	13.66	25.0
Slime tail	50	25	1.2	33.4	1.71	30.0	44.98	31.7
Total	200	100	.....	94.6	.....	101.6	.....	98.1

Flotation without the use of chemicals gave encouraging results. Sodium sulphide was then used to sulphidize the metallic oxides preparatory to flotation (see Tables 30, 31, and 32). Five hundred grams of minus 100-mesh ore was ground in a porcelain jar with iron balls for 20 minutes with 2 grams of sodium sulphide (fused, 60 per cent) and 10 drops of a mixture of 3 parts coal tar to 1 part cresylic acid. The charges were floated in a modified M. S. machine with the addition of 5 drops G. N. S. No. 5 oil and four drops of P. T. & T. No. 350 oil.

TABLE 30.—*Flotation of Senorita ore after treatment with sodium sulphide.*

Product.	Weight of products (grams).	Per cent of total weight.	Lead.		Copper.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	500	100	0.6	100.0	1.06	100.0	26.10	100.0
Concentrate	100	20	2.3	78.7	3.35	63.2	111.50	85.5
Tailing	400	80	.20	26.6	.40	30.2	4.67	14.3
Total	500	100	.....	103.3	.....	93.4	.....	99.8

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TABLE 31.—*Flotation of Black Girl ore after treatment with sodium sulphide.*

Product.	Weight of products (grams).	Per cent of total weight.	Lead.		Copper.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	500	100	0.50	100.0	0.96	100.0	24.48	100.0
Concentrate	105	21	2.00	84.0	3.76	82.4	110.00	94.5
Tailing	395	79	.10	15.8	.23	19.0	1.62	5.2
Total	500	100	.....	99.8	.....	101.4	.....	99.7

TABLE 32.—*Flotation of Wedge ore after treatment with sodium sulphide.*

Product.	Weight of products (grams).	Per cent of total weight.	Lead.		Copper.		Silver.	
			Assay (per cent).	Per cent of feed content.	Assay (per cent).	Per cent of feed content.	Assay (ounces per ton).	Per cent of feed content.
Feed	500	100	0.50	100.0	0.14	100.0	32.10	100.0
Concentrate	70	14	2.5	70.0	.50	50.0	229.5	73.5
Tailing	430	86	.2	34.2	.08	49.1	9.7	26.0
Total	500	100	.....	104.2	.....	99.1	.....	99.5

Undoubtedly, the use of sodium sulphide is advantageous in the flotation of these ores and the manner of application should be studied. As sodium sulphide oxidizes rapidly, its addition to the charge just before flotation begins might give better results.

ROBERT E. LEE AND DUNKIN, AT LEADVILLE.

The ore from the Robert E. Lee mine assayed 9.18 ounces silver, and that from the Dunkin mine, 4.92 ounces silver.

Tests made by agitating 100-mesh material with air in cyanide solution for 24 hours gave the following results:

Results of cyanidation tests of Robert E. Lee and Dunkin ores.

Ore.	Extraction of silver.	NaCN consumed per ton of ore.	CaO consumed per ton of ore.
	Per cent.	*Pounds.*	*Pounds.*
Robert E. Lee	78.7	1.5	3
Dunkin	75.0	1.5	3

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Ore ground to pass a 200-mesh sieve showed no higher consumption of cyanide and lime, but gave a slightly higher extraction of the silver—81 per cent for the Robert E. Lee and 75.5 for the Dunkin. Increasing the time of agitation to 96 hours gave no higher extraction.

Cyaniding followed by sulphidizing and flotation would probably give very low-grade tailings.

COMMENTS ON FOREGOING TESTS OF OXIDIZED ORES.

Eleven oxidized ores were tested. Most of the work, naturally, was cyaniding; a few flotation tests were made. The equipment and the time available permitted only the most cursory examination. There was no opportunity for testing the effectiveness of the "working" solutions; although when cyanide consumption is as high as in some of the tests, the solution becomes polluted and inactive. Flotation has the advantage that solutions do not have to be used repeatedly to conserve reagents. The lavish use of abundant fresh water is of advantage in flotation, whereas it would cause much loss in cyanidation. Consequently, in choosing between flotation and cyanidation, the quantity of the available water is an important factor.

The results are encouraging, and justify plans for elaborate tests of those ores of which the available tonnage is adequate.

In all, 29 ores were examined and their amenability to concentration determined as far as time allowed. Doubtless other laboratories have done equivalent work, but the results have not been published by the participating companies. Such failure to publish results by private concerns supplying the funds is no breach of ethics, but publication should be encouraged in the interest of the public.

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